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Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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Industrial Section . . . . (Interpolated in Advertising Section)

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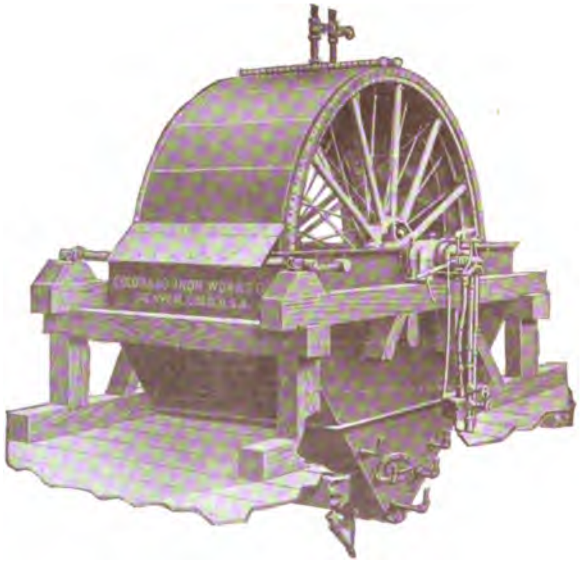
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(Name in Full)

Occupation ______________________________

Address ______________________________

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____

of the American Institute of Mining Engineers.

____
____
____
Signatures of three Members or Associates.

Place of birth ______________________ *Year of birth* ________

Education, general and technical, when, where and how acquired, with degrees, if any.

Dates	

Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)

Dates	

Present position:

(If the applicant has previously been proposed as a Member, Associate or Junior Member of the Institute, this fact should be stated.)

I agree, if elected, to accept election within four months, and to abide by the Constitution and By-Laws

Signature of Applicant:

Dated *191*

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

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Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

SPECIAL TRAIN TO THE ARIZONA MEETING

As already announced, the Arizona Meeting of the Institute is to be held from the 18th to the 25th of September. There will be a special train from New York City to Arizona and return, to take members and guests of the Institute to the meeting. The Committee in charge consists of Walter Douglas, Chairman, Cleveland E. Dodge, John C. Greenway, Arthur S. Dwight and Julius Kruttschnitt, Jr., and while the plans of this Committee are not complete, it may be announced that the train will be run with the requisite number of Pullman, dining and baggage cars to accommodate those who desire to go, which will accompany the party during the entire week of the Arizona Meeting as per the following program, including, of course, the visit to the Grand Canyon.

As soon as details have been worked out, a circular will be issued giving full information concerning the trip.

ARIZONA MEETING, SEPTEMBER 18 to 24, 1916

The Arizona Committee, consisting of

GERALD F. G. SHERMAN, *Chairman*,
ARTHUR NOTMAN, *Secretary*,
NORMAN CARMICHAEL, B. BRITTON GOTTSBERGER,
W. L. CLARK, J. C. GREENWAY,
W. G. MCBRIDE, FOREST RUTHERFORD,

has arranged the following program for the Arizona Meeting, which has been approved by the President of the Institute and by the Executive Committee. It is printed herewith for the information of members, and probably will not be changed in any material respect.

Sunday, September 17, 1916.—Arrive at El Paso, Texas, 2.40 P.M., leave 10.20 P.M.

Monday, September 18, 1916.—Arrive at Deming, N. Mex., 12:47 A.M. Arrive at Santa Rita, N. Mex., 4.00 A.M., where luncheon will be served, after which there will be a visit to the Chino mines. From there the train will proceed to Hurley, N. Mex., where a visit will be made to the mill of the Chino Copper Co.

Tuesday, September 19, 1916.—Arrive at Douglas, Ariz., at 8.00 A.M. The morning will be spent at the reduction works of the Copper Queen Consolidated Mining Co. and of the Calumet & Arizona Mining Co. After luncheon a technical session will be held, with "Smelting" as a special subject for discussion. There will be a banquet in the evening, followed by another technical session or a business meeting.

Wednesday, September 20, 1916.—Special train will arrive at Bisbee, Ariz., at 7.00 A.M., and the morning will be devoted to visiting the mines of the district. After luncheon, a technical session will be held with "Mining and Geology" as the special subject of discussion. In the evening there will be a banquet at the Warren District Country Club, followed by a social evening; the special train leaving at 10.00 P.M.

Thursday, September 21, 1916.—Arrive at Globe, Ariz., at 9.00 A.M. The morning will be spent in visits to the mines of the Old Dominion Copper Co. After luncheon a technical session on the subject of "Mining and Smelting" will be held and, after a banquet in the evening, there will be a technical session on the subject of "Leaching."

Friday, September 22, 1916.—In the morning visits will be made to the reduction works of the Old Dominion Copper Co., the International Smelting & Refining Co., and the mill of the Miami Copper Co. After luncheon, the subject of the technical session will be "Concentration and Flotation." A banquet will be held, followed by a social evening.

Ray.—Members desiring to visit Ray, Ariz., may leave by automobile in the afternoon, visiting mines the next morning and reaching Phoenix in time to join the party going to the Grand Canyon.

Saturday, September 23, 1916.—Those who do not take the Apache Trail trip will arrive in the special train at Phoenix, Ariz., at 6.00 P.M. The special train will leave for the Grand Canyon at 8.00 P.M.

Apache Trail Trip and Roosevelt Dam.—Members desiring to take the Apache Trail trip including a visit to the Roosevelt Dam will leave Globe by automobile on the morning of Saturday, reaching Phoenix that evening.

Jerome.—Members desiring to visit Jerome should communicate with the Secretary in order that special arrangements may be made at Jerome and meet the special train for the Grand Canyon at Jerome Junction.

Sunday, September 24, 1916.—Arrive at El Tovar, Grand Canyon, at 8.00 A.M. All day Sunday will be available to those members of the party who desire to take some of the famous horseback rides into the Grand Canyon or rides or drives along the rim to various points of interest.

NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President:
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

> "The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.
>
> District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N. B.—New York City and district is designated District 0.)
> District No. 2. Pennsylvania.
> District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
> District No. 4. Minnesota, Wisconsin, and Michigan.
> District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
> District No. 6. California and Nevada.
> District No. 7. Utah, Colorado, Arizona, and New Mexico.
> District No. 8. Louisiana and Texas.
> District No. 9. Other Southern States and District of Columbia.
> District No. 10. Mexico.
> District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

> "In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:
President, L. D. Ricketts (not eligible for re-election).
Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
Chairman, Committee on Nominations.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, MAY 26, 1916

The following members were appointed on the Committee Advisory to the U. S. Bureau of Standards:

Arthur S. Dwight (Lead).
John Fairfield Thompson (Nickel).

The reports of the Committee in charge of the Arizona Meeting and the Committee on Transportation to the Arizona Meeting were accepted with the thanks of the Board.

The By-Laws of the Nevada Section of the Institute were approved.

The subscription price of the Bulletin to non-members of the Institute was increased to $12 per year beginning with January, 1917. This increase does not affect the reduced price of $5 on extra copies of the Bulletin which are sold to members, libraries and sister societies.

Mr. Sidney J. Jennings was appointed delegate of this Institute on a Committee of Conference to act with the National Academy of Science.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period May 10, 1916 to June 10, 1916:

G. Aertsen, Philadelphia, Pa.
Frederick C. Alsdorf, Wickenburg, Ariz.
A. H. Anderson, Worcester, Mass.
J. H. Batcheller, Mattapoisett, Mass.
Arthur B. Clark, Sheridan, Mont.
J. Murray Clark, Toronto, Canada.
J. H. Collier, Oakland, Cal.
H. S. Emlaw, Spring Lake, Mich.
E. L. Estabrook, Platteville, Wis.
James W. Hambleton, El Paso, Tex.
James Henderson, London, England.
G. P. Hulst, East Chicago, Ind.
William F. Jahn, Riebeling, Mont.
Hennen Jennings, Washington, D. C.
H. M. LaFollette, LaFollette, Tenn.
E. T. Lednum, Chicago, Ill.
Francis Church Lincoln, Reno, Nev.
H. Mortimer-Lamb, Montreal, Canada.
Andrew W. Newberry, San Francisco, Cal.
C. E. Pettibone, Cleveland, O.
Heinrich Ries, Ithaca, N. Y.
C. S. Robinson, Youngstown, O.
M. K. Rodgers, Basin, Mont.
E. J. Rossbach, St. Louis, Mo.
J. B. Tyrrell, Toronto, Canada.
G. B. Waterhouse, Buffalo, N. Y.

H. S. Kimball has been elected President of the Aetna Explosives Co., Inc.

A. J. Moxham, formerly president of the Aetna Explosives Co., Inc., has become chairman of the board.

A. D. Little has been appointed head of a bureau to co-ordinate the work of scientific research in Canada at Montreal.

George J. Young, formerly director of the Mackay School of Mines, Reno, Nev., has been elected Professor of Metallurgy at the Colorado School of Mines beginning September, 1916.

Arthur J. Hoskin has resumed partnership with Frank J. McNeir, with headquarters at Leadville, Colo.

Bertram D. Coleman has resigned his position as a Director of the Pennsylvania Steel Co. and of the Cornwall Ore Banks Co.

H. C. Wilmott has joined the staff of the Andes Exploration Co., Santiago, Chile.

George E. Farrish has been appointed Consulting Engineer for the Tri-Bullion Smelting & Development Co. of New York.

Samuel M. Vauclain has been elected a director of the Westinghouse Electric & Manufacturing Co.

Andrew W. Newberry and **Dr. Francis Church Lincoln,** Director of the Mackay School of Mines, University of Nevada, Reno, Nev., sailed from New York City on May 27, bound for Peru and Bolivia on mining business.

N. H. Darton has returned from Cuba, where he has been investigating prospects for artesian water on one of the large sugar plantations near Guantanamo.

E. B. Durham has joined the engineering staff of the Mammoth Copper Mining Co., Kennett, Cal., and is engaged in the design and erection of an aerial tramway at the Stowell Mine, near Kennett.

E. I. Montoulieu, of the Department of Public Works of the Government of Cuba, is at the United States Mint at Philadelphia in the capacity of Technical Expert and Inspector for his Government, in connection with the new coinage of the Republic. From March, 1915, when the coinage was started, to date, a total of 62,474,740 pieces of gold, silver and nickel, valued at $29,952.203 U. S. currency, have been shipped to Havana.

Robert J. Anderson has accepted a position as metallurgist with the American Rolling Mill Co., Middletown, Ohio.

Lyon Smith has accepted appointment as metallurgist for the Snyder Electric Furnace Company with headquarters at Chicago, Ill.

G. H. Cunningham, Superintendent of Construction for Zinc Electrolytic Plants, Washoe Reduction Works, Anaconda, Mont., has recently been appointed Construction Engineer in charge of installation of equipment in the new 100 ton Zinc Electrolytic Plant which the Anaconda Copper Mining Co. is building at Great Falls, Mont.

Albert A. Leach has been promoted to Chief Geologist of the Burro Mountain Copper Co., at Tyrone, New Mexico, a Phelps-Dodge Corporation.

POSITIONS VACANT

A steel manufacturing company is desirous of securing an assistant superintendent for its plant. Experience in open-hearth plants and in rolling mills particularly desirable. No. 112.

Opening as assistant superintendent of a non-ferrous smelting plant. Man with metallurgical experience preferred but chief essential is ability to handle men. No. 114.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, aged 41, mining engineer, manager or general superintendent, 20 years' experience, 10 years in Mexico, 5 years Western States, 5 years abroad. Guarantee results, fluent Spanish, excellent references. New York interview. No. 282.

Member, technical mining graduate, 10 years' experience, especially in design, erection, construction and purchase of power and mining equipment and supplies, desires position in purchasing department. Married, aged 32, employed at present, references furnished. No. 295.

Member, technical graduate, B.S. and E.M. degrees. Ten years' experience in metal and coal mines and metallurgical plants. Open for engagement as superintendent or manager. Available at 30 days' notice. Interview in Chicago, New York, or Pittsburgh. No. 296.

Member, aged 33, broad technical education and 12 years' general mining experience with especial experience in preliminary examinations, wishes position with examining engineers or exploration company. Salary not first consideration. No. 297.

Member, B.S. in Chemistry, aged 29, now and for the past seven years with large custom copper smelter, expert in control assaying of all kinds of ores and products of copper and associated metals. Has had general experience with blast furnaces, reverberatories, roasters, converters and sampling works in preparation for position as assistant superintendent. Health and habits O.K. Fluent Spanish, fair French and German. Married. No. 298.

Member, Cornell M. E. graduate, aged 31, married. Will soon complete two years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15th in the United States. No. 299.

LOCAL SECTION NEWS

NEVADA SECTION

J. W. HUTCHINSON, *Chairman*,
FRANCIS CHURCH LINCOLN, *Vice-Chairman*,
HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada
W. H. BLACKBURN, E. A. JULIAN,
EMMET D. BOYLE, JOHN G. KIRCHEN,
FREDERICK BRADSHAW, C. B. LAKENAN,
TASKER L. ODDIE.

The formation of the Nevada Section was originally suggested by Professor F. C. Lincoln, who had discussed the subject with Mr. William L. Saunders when the latter was President of the Institute. Professor Lincoln sought the assistance of the Nevada Mine Operators' Association, whose Executive Committee constituted itself a Committee on Organization, and authority for the establishment of a Nevada Section to include the entire State was secured from the Board of Directors of the Institute.

The formal installation took place at 10 A.M. on May 18, Mr. William L. Saunders officiating. The Honorable Emmet D. Boyle, Governor of Nevada (and a member of the Nevada Section), made the address of welcome.

On the afternoon of the same day, a technical session was held, at which the following papers were presented:

History and Development of the Cyanide Process	Charles Butters
Virginia City	Whitman Symmes
Ore Deposition at Steamboat Springs	Prof. J. C. Jones

At noon on the 18th, the Nevada Section tendered a luncheon to the ladies accompanying members, this luncheon being in charge of Mrs. Emmet D. Boyle, Mrs. Francis C. Lincoln and Mrs. Henry M. Rives, and in the evening the first banquet of the Nevada Section was held. At the conclusion of the banquet, Professor Lincoln delivered an illustrated lecture on The Evolution of Mining.

On the 17th, occurred the Commencement Exercises of the University of Nevada, a Commercial Club luncheon in honor of the Hon. Charles A. Towne, the reception to the friends and guests of the University at the residence of President Hendrick, and the Alumni-Senior Ball. At this ball, the members and ladies were guests of the Alumni Association of the University.

On the 19th, the party motored to Virginia City, where an opportunity for seeing the Mexican, Consolidated Virginia, and other mines, was offered through the courtesy of Mr. Whitman Symmes and Mr. T. F. McCormick. On the return trip, a stop was made at Carson, where the party was received at the Governor's Mansion by Governor and Mrs. Boyle.

CHICAGO SECTION

CHARLES H. MACDOWELL, *Chairman,*
LUTHER V. RICE, *Vice-Chairman,*
HENRY W. NICHOLS, *Secretary-Treasurer,* 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

The President and the Secretary of the Institute were guests of the Chicago Section at its dinner and meeting of May 22.

The talks and discussions after the dinner clung persistently to the subject of conservation of mineral resources.

Dr. Ricketts told of the great economies effected by the copper smelters in recent years, and showed that similar methods can be applied to the other metallurgical industries.

Mr. Bradley Stoughton told of the Institute's interest in its sections and of its plans for increasing the importance of the sections.

Mr. F. L. Peabody, of the Peabody Coal Co., spoke of the great waste in coal mining in this country. He said that the average extraction of coal in Germany is 97½ per cent. as compared with about 60 per cent. in Illinois. He said that to permit a high extraction there must be a Federal law compelling the use of a retreating system for coal mining, as it would be impossible for mines operated on this system to compete with mines operated along the present lines. Mr. Peabody showed moving pictures of both the above-ground and the underground operation of two large Illinois coal mines, one of which hoists 8000 tons through one shaft in eight hours.

The discussion of the papers showed that the section was heartily in favor of the kind of conservation advocated by the speakers; that is, not the withholding from use, but the avoidance of all waste when extracting and using.

There were 47 guests and members present.

H. W. NICHOLS, *Secretary.*

ST. LOUIS SECTION

C. J. ADAMI, *Chairman,*
HERMAN GARLICH, F. W. DE WOLF, M. M. VALERIUS, *Vice-Chairmen,*
W. E. MCCOURT, *Secretary-Treasurer,* Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR,
C. R. FORBES, F. D. RASH,
ARTHUR THACHER.

The annual meeting of the St. Louis Section was held at the St. Louis Club on May 20, 1916. The meeting was preceded by a dinner, at which were present 53 members and guests, including Dr. L. D. Ricketts, President of the Institute, and Mr. Bradley Stoughton, Secretary. Mr. Philip N. Moore presided at the meeting.

Dr. Ricketts addressed the Section on the subject of The Present Condition and Outlook of the Copper Industry of the West, and Mr. Stoughton on the subject of Local Sections.

The members elected the Executive Committee for the ensuing year as shown above.

About 35 members spent the day visiting the coke plant of the Laclede Gas Light Co. in St. Louis, and the works of the Hoyt Metal Co., and the National Stamping and Enameling Company of Granite City, Illinois.

The membership of the Section is now 184.

W. E. McCOURT, *Secretary.*

PUGET SOUND SECTION

GLENVILLE A. COLLINS, *Chairman,*
H. L. MANLEY, *Vice-Chairman,*
AMOS SLATER, *Secretary-Treasurer,* 1043 Henry Bldg., Seattle, Wn.
JOHN N. POTT, I. F. LAUCKS.

The monthly meeting of the Puget Sound Section was held at the Arctic Club, May 18, 1916, at 6.30 P.M.

After dinner had been served, the meeting was called to order by the Chairman.

The Chairman suggested that this section hold a Mining Convention this coming winter. This was placed in form of a motion and carried. A committee of three—Chester F. Lee (Chairman), I. F. Laucks and Joseph Daniels—was appointed to investigate and arrange as far as possible, and to report progress to the Executive Board.

Milnor Roberts outlined the work of the Bureau of Mines in establishing branches in the different States, and suggested that the Puget Sound Section assist in obtaining a station for the University of Washington. This was placed in the form of a motion, and after discussion the motion was carried.

The Chairman called upon Capt. A. O. Powell, Chairman of the Associate Naval Consulting Board of the United States, to explain the work of this Board in relation to the State of Washington. Capt. Powell responded, outlining the work of the Board and telling how they expect to accomplish such work.

The Chairman then introduced the speaker for the evening, Mr. John C. Ralston, of Spokane, who delivered an address on the Catskill Water Supply of New York City, using lantern slides for illustration. This address was exceedingly interesting, and called forth discussion at the end of the lecture.

The Chairman thanked Mr. Ralston in behalf of the Section, and the meeting adjourned.

AMOS SLATER, *Secretary.*

SAN FRANCISCO SECTION

T. A. RICKARD, *Chairman,*
W. H. SHOCKLEY, *Vice-Chairman,*
C. E. GRUNSKY, JR., *Secretary-Treasurer,* 57 Post St., San Francisco, Cal.
E. A. HERSAM, H. W. YOUNG.

On May 31, 1916, the San Francisco Section gave a dinner in the rooms of the Engineers' Club. A number of the members of the San Francisco Association of the American Society of Civil Engineers were present in response to an invitation extended to them.

Dr. John Casper Branner, member of the Commission of the National Academy of Sciences which recently visited the Isthmus of Panama at the request of President Wilson, delivered an address on The Panama Canal Slides. He called attention to the dangerous conditions on the Isthmus which may lead to further movement of these large earth masses. A discussion which was participated in by C. E. Grunsky (A. S. C. E.), George L. Dillman (A. S. C. E.), Dr. A. C. Lawson, and others, brought out many details of the history of the Canal slides. It was pointed out that slides were already in evidence in Culebra Cut at the time the French withdrew and the United States took up the task of completing the Canal.

C. E. GRUNSKY, JR, *Secretary.*

AFFILIATED STUDENT SOCIETY NOTES

The Mining and Geological Society of Lehigh University has elected the following officers for the ensuing year.

R. L. MCCANN, *President,*
ALAN E. DYNAN, *Secretary.*

The Society was fortunate in securing for the April and May meetings the following speakers, who gave addresses on the subjects indicated:

Prof. E. S. Moore of Pennsylvania State College, on Mining Conditions in Australia and India.

Mr. D. S. Chamberlain of the Lehigh Coke Company, on What Comes from a Ton of Coal.

ALAN E. DYNAN, *Secretary.*

The **Student Branch** at the **Ohio State University** has held a number of meetings during the past few months, and has listened to the following talks:

India and the Far East. By Mr. J. K. Bowden.
Bauxite Deposits in the Southern U. S. By Mr. E. H. Shriver.
Steel Making at Gary. By Mr. E. C. Smith.
Natural Gas. By Mr. E. C. Ramsey.
A Summer in Bingham Canyon. By Walter Carroll.

The Branch gave a dance in March and has organized a string orchestra and a baseball team. We hope for a bigger and better society next year.

We are dependent to a large extent on visiting Alumni for our talks, and this makes impossible any fixed program.

W. J. POUCHOT, *President.*

The Illinois Student Branch closed its work for the year by having two smokers. These were held at fraternity houses upon invitation of members of the society, and were well attended.

At the first smoker an informal discussion was held upon the subject of Mine Safety. At the second meeting the subject of discussion was Mining Finance. Formal papers were not attempted, but the informal discussions were very generally participated in, so that the two closing meetings were probably the most successful held throughout the year.

As one result of the Safety First discussion, the members expressed a desire to establish in the fall a voluntary class for systematic first-aid instruction, looking toward the securing of certificates from the National Red Cross, the United States Bureau of Mines or the Mine Rescue Station Commission of the State of Illinois.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining

COAL MINE FATALITIES IN THE UNITED STATES, 1915. List of permissible explosives, lamps and motors tested prior to January 1, 1916. Washington, 1916.

EXPLOSIBILITY OF GASES FROM MINE FIRES. Tech. Paper 134, U. S. Bureau of Mines. Washington, 1915.

EXPLOSIVES FOR SHALE AND CLAY BLASTING. Wilmington, 1916. (Gift of E. I. du Pont de Nemours & Co.)

INSTRUMENTS FOR RECORDING CARBON DIOXIDE IN FLUE GASES. Bull. 91, U. S. Bureau of Mines. Washington, 1916.

INFLAMMABILITY OF ILLINOIS COAL DUSTS. Bull. No. 102, U. S. Bureau of Mines. Washington, 1916.

MINE VENTILATION STOPPINGS, WITH ESPECIAL REFERENCE TO COAL MINES IN ILLINOIS. Bull. No. 99, U. S. Bureau of Mines. Washington, 1915.

PRACTICAL SHAFT SINKING. By Francis Donaldson. New York, 1910.

Metallurgy and Chemistry

CHEMISCHE TECHNOLOGIE. By Fr. Heusler. Leipzig, 1905.

DIE CHEMIE DER KOHLE. By F. W. Hinrichsen & S. Taczak. Ed. 3. Leipzig, 1916.

FOUNDRYMAN'S REFERENCE BOOK. By J. F. Bowe. Pittsfield, n.d.

HISTORY OF THE IRON, STEEL, TINPLATE AND OTHER TRADES OF WALES. By Charles Wilkins. Merthyr Tyofil, 1903.

METALLURGY OF GOLD. Ed. 6. By T. K. Rose. London, 1915.

STEEL, AND ITS HEAT TREATMENT. By Denison K. Bullens. New York, J. Wiley & Sons, 1916. (Gift of Publishers.)

[NOTE.—This work takes up the heat treatment of steel for annealing, hardening, tempering and case carburizing. It has chapters on special steels and their treatment, and gives descriptions of appropriate apparatus. W. P. C.]

THE FLOTATION PROCESS. Compiled and edited by T. A. Rickard. San Francisco, 1916.

Geology and Mineral Resources

COAL RESOURCES OF DISTRICT VIII (DANVILLE). Bull. No. 14, Illinois Coal Mining Investigations. Urbana, 1915.

GEOLOGY AND GEOGRAPHY OF THE GALENA AND ELIZABETH QUADRANGLES. Bull. No. 26, Illinois State Geological Survey. Urbana, 1916.

GREAT BRITAIN. MINES AND QUARRIES: GENERAL REPORT WITH STATISTICS FOR 1915. Advance proof of the tables relating to the output of coal and other minerals and the number of persons employed at mines worked under the Coal and Metalliferous Mines Acts during 1915. London, 1916.

THE ANTHRACITE COAL FIELDS OF PENNSYLVANIA, AND THEIR EXHAUSTION. Harrisburg, 1881.

Military Engineering

CONSTRUCTION OF IMPROVED ORDNANCE. By Daniel Treadwell. Cambridge, 1862.

MILITARY AEROPLANES. By G. C. Loening. Ed. 2. Boston, 1916.

MODERN GUNS AND GUNNERY, 1910. By H. A. Bethell. Woolwich, 1910.

General

APPLETONS' NEW SPANISH-ENGLISH AND ENGLISH-SPANISH DICTIONARY. By Arturo Cuyás. New York, 1916.

HANDBOOK OF FORMULAS AND TABLES FOR ENGINEERS. Compiled by Clarence A. Perice and W. B. Carver. New York, 1914.

HANDBUCH DER RADIOLOGIE. By Erich Marx. Band III. Leipzig, 1916.

HARPER'S HYDRAULIC TABLES FOR THE FLOW OF WATER. By J. H. Harper. New York, 1916.

STRAIGHT LINE ENGINEERING DIAGRAMS. By Manifold and Poole. San Francisco, 1911.

Company Reports

WALLAROO AND MOONTA MINING AND SMELTING COMPANY, LTD. Annual Report 26th Adelaide, 1915. (Gift of H. L. Hancock.)

Trade Catalogs

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bull. No. 192. Stone tools, stone dressers, air compressors and equipment for stone. April, 1916.
Form 249. Chicago Portable Mine Hoist, Photographs of.

DIAMOND POWER SPECIALTY CO., Detroit, Mich. Power Notes. April, 1916.

INGERSOLL-RAND CO., New York, N. Y.
Form 3026. "Ingersoll-Rogler" Class "PRE" duplex direct-connected, electrically driven air compressors. March, 1916.
Form 3312. "Imperial XB" duplex power-driven air compressors. Nov., 1915.
Form 9023. "Imperial" Tie tamping outfits. March, 1916.

LESCHEN & SONS ROPE CO., St. Louis, Mo. Coal mining in southern Indiana. May, 1916.

NATIONAL TRANSIT PUMP AND MACHINE CO., Oil City, Pa. Bull. 405. Horizontal gas engines.

NATIONAL TUBE CO., Pittsburgh, Pa. Bull. No. 25. "National" pipe in large buildings.

PENSACOLA TAR AND TURPENTINE CO., Gull Point, Fla. Description of pine flotation oils.

A BIBLIOGRAPHY ON VALUATION OF PUBLIC UTILITIES

The Committee on Valuation of the American Electric Railway Association, has had compiled a "Bibliography on Valuation of Public Utilities." The work was done by the library force of the American Society of Civil Engineers, and brings up to Dec. 23, 1915, a bibliography issued by the American Society of Civil Engineers on May 15, 1912.

The Bibliography has been printed and issued by the American Electric Railway Association, 8 West 40th Street, New York, N. Y., and may be obtained from its Secretary on the payment of 50 c.

References are arranged under the following headings: General, Electric Light and Power, Gas Works, Mining, Railroads, Steam Power, Street and Inter-urban Railways, Telegraph and Telephone, Water Power, Water Works.

References are given to the principal technical publications, to various works on the question of valuation, to rules and regulations of Commissions and other official bodies, and to the proceedings of various societies.

Undoubtedly the compilation is the most comprehensive that has yet been published.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period May 10, 1916, to June 10, 1916:

ARCHIBALD, JOHN CHRISTIE, Min. Engr., Supt. of Cyanide Plant, Guanajuato Reduction Mines, Guanajuato, Mex.
BRUNEL, FRANK P., Cons. Min. Engr............... Mitchell & Brunel, Ajo, Ariz.
CALVERT, WILLIAM ROBERT, Cons. Geol. ...225 Kearns Bldg., Salt Lake City, Utah.
CHAMPION, J. R., Supt.....The Yak Mining, Milling & Tunnel Co., Leadville, Colo.
CHARLES, LAVERN JOHN, Construction Engr., Elephant Butte Dam, Elephant Butte, N. M.
CHASTEL, ANDRE, Min. Engr., Societe Miniere & Metallurgique de Penarroya, 12, Place Vendome, Paris, France.
CHRISTIE, WALLACE H., Vice-Pres. and Mgr., Alta Bert Gold Dredging Co., 604 Mission St., San Francisco, Cal.
CLARK, RONALD.....Hardinge Conical Mill Co., 120 Broadway, New York, N. Y.
CROSLEY, FRED W.................408 E. Bennett Ave., Cripple Creek, Colo.
D'ARCY, ARTHUR I., Min. Engr., Atlanta Mines Co., P. O. Box 1775, Goldfield, Nev.
DAVENPORT, JOHN, Min. Engr., Supt., United Mineral Co., So. Danbury, N. H.
DEVER, P. H., Asst. Dist. Supt., Delaware, Lackawanna & Western R. R. Co., Nanticoke, Pa.
DOWNES, FRANK A., Assayer...................Aurora Cons. Mines, Aurora, Nev.
FRY, LOUIS D., Min. Engr., Copper Queen Cons. Min. Co., P. O. Box 1358, Bisbee, Ariz.
GILL, ARTHUR HENDRIX........Supt, American Smelt. & Ref. Co., Maurer, N. J.
VAN WATERSCHOOT VAN DER GRACHT, W. A. I. M., Geol. and Min. Engr., Care Roxana Petroleum Co., Tulsa, Okla.
GRIFFITH, JOHN R., Research Engr..............Norton Co., Niagara Falls, N. Y.
GRISWOLD, GEORGE GARY, Supt., Smelting & Lead Refining, American Smelt. & Ref. Co., Maurer, N. J.
GUERNSEY, JOHN B., Reorganization Mgr., The Low Moor Iron Co. of Va., Low Moor, Va.
HARMS, PURL L., Mine Sampler...........Homestake Mining Co., Lead, So. Dak.
HARPER, HARRY ARTHUR, Min. Engr., Braden Copper Co., Rancagua, Chile, S. A.
HENRIE, H. C., Chief Chemist.........Copper Queen Cons. Min. Co., Bisbee, Ariz.
JARVIS, WILLIAM R., Dist. Mgr., Sullivan Machinery Co. (Chicago, Ill.), 1513 Farmers Bank Bldg., Pittsburgh, Pa.
JONES, THOMAS B., Chief of Spelter Operation, Mineral Point Zinc Co., Depue, Ill.
KERNS, RALPH W., Engr. Smelting Plant Constr., New Cornelia Copper Co., Warren, Ariz.
KIVARI, ARTHUR M., Chief Mill Draftsman, Vindicator Cons. Gold Min. Co., Box 495, Victor, Colo.
LEWIS, JAMES BRADFORD, JR., Min. Engr., Calumet and Arizona Mining Co., Warren, Ariz.
MCDONALD, JOHN A., Engr., Testing Dept., Washoe Smelter, Anaconda Copper Mining Co., Anaconda, Mont.
MCNAB, ALEXANDER JOHN, Mgr.......Mason Valley Mines Co., Thompson, Nev.
MACGREGOR, FRANK S..........................60 India St., Boston, Mass.
MERRYWEATHER, HUBERT, Vice-Pres. and Genl. Mgr., Bethlehem Chile Iron Mines Co., La Higuera, Coquimbo, Chile, S. Amer.
MOULTON, JOHN CARROLL, Mill Supt., Pittsburg Silver Peak Gold Min. Co., Blair, Nev.
NAJARIAN, HERAND K., Engr. on Steam Shovel Operations, Nevada Cons. Copper Co., Ruth, Nev.
NELSON, WILLIAM INNES, Millman, Alaska Gastineau Mining Co., Thane, Alaska.
NEWHALL, PERCY MELROSE, Asst. Cons. Eng., S. Neumann & Co., P. O. Box 485, Johannesburg, S. Africa.
NITCHIE, CHARLES C., Chemical Engr............Mineral Point Zinc Co., Depue, Ill.
NORTH, MORTIMER S., Min. Engr.....Calumet & Hecla Mining Co., Calumet, Mich.
PARISH, RALPH RAND, Cons. Met........Rome Brass & Copper Co., Rome, N. Y.

Partanen, Isak, Min. Engr..The Colorado-Superior Min. Co., Telluride, Colo.
Peterson, Clarence J., Min. Engr........Tonopah Mining Co., Tonopah, Nev.
Prior, Charles Edward, Assayer and Chemist, Daly Reduction Co., Hedley, B. C., Canada.
Queen, Edward L., Mining........................White Oaks, N. Mex.
Rosenbaum, Rudolph R., Chemical Engr., Care Caddo Oil Refinery, Shreveport, La.
deRyck van der Gracht, C. M. A., Managing Director, The Petroleum Maatschappy Saltcreek, 22, Lange Houtstraat, The Hague, Holland.
Samuel, John Moore, Testing Engr., Reduction Works, Copper Queen Cons. Min. Co., Douglas, Ariz.
Sharp, William, Min. Engr., Wingfield Outside Properties, Box 597, Goldfield, Nev.
Smith, Malcolm M., Min. Engr., The Virginia Coal & Iron Co., Big Stone Gap, Va.
Stevens, William Leonard, Met., Met. Accountant, Braden Copper Co., Rancagua, Chile, S. A.
Taylor, George M., Mgr., Mill. Dept., Portland Gold Mining Co., Colorado Springs, Colo.
Tenney, James B., Min. Geol., Geol., Copper Queen Cons. Min. Co., Warren, Ariz.
Terrell, Arthur Davis, Genl. Mgr., Mineral Point Zinc Co., 1111 Marquette Bldg., Chicago, Ill.
Thomas, Richard B., U. S. Mineral Surveyor...............Colville, Wash.
Tyler, Paul McIntosh, Min. Engr.......................Breckenridge, Colo.
Walker, Edwin Harold, Min. Engr., Underground Foreman, Utah Copper Co., Bingham Canyon, Utah.
Wasson, H. J., Min. Engr., Pittsburg-Dolores Mining Co., Rockland, via Yerington, Nev.
Winans, Henry S., Selling Engr., The W. S. Tyler Co., 1212 Josephine St., Denver, Colo.
Wray, Daniel C., Asst. Supt., Nassau Plant, Mineral Point Zinc Co., Depue, Ill.

Associate Members

Thomas, Richard P., Asst. Dist. Supt., Delaware, Lackawanna & Western R. R. Co., Kingston, Pa.

Junior Members

Annett, Charles L., Min. Engr..........100 Warm Springs Ave., Boise, Idaho.
Burg, Robert Stanley, Student..........Missouri School of Mines, Rolla, Mo.
Kernan, Thomas H., Student...............Univ. of Wisconsin, Madison, Wis.
Kluge, Harry A., Student....Missouri School of Mines and Metallurgy, Rolla, Mo.
Maxwell, Norman E., Student.............................Golden, Colo.
Noon, Thaddeus Roderick...........................1322 4th St., Peru, Ill.
Porter, Fred S.....Care Alaska Treadwell Gold Mining Co., Treadwell, Alaska.
Roberts, Dudley E., Student..............18 Pleasant St., Stamford, Conn.
Skirm, Joseph Gaskill, Student.........University of Wisconsin, Madison, Wis.
Weiberg, Earl B., Student, Missouri School of Mines and Metallurgy, Rolla, Mo.

Total Membership, June 10, 1916........................ 5,537

CANDIDATES FOR MEMBERSHIP

Application for Membership.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period May 10, 1916 to June 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Louis M. Allen, Jr., Globe, Ariz.
Proposed by I. H. Barkdoll, H. L. Norton, W. B. Cramer.
Born 1887, New York City. 1905–07, 1908–09, Colorado School of Mines, Golden, Colo. 1907, Furnace man, U. S. Metals Refining Co., Chrome, N. J. 1908, Underground, Tonopah Min. Co. 1909, Surveying for Mr. A. H. Koebig, Los Angeles, Cal. 1910, Asst. Engr., in examination work in Mexico for Mr. Frank A. Keith, of Los Angeles, Cal. 1914, Engineer's office and Assayer, Nevada Cons. Copper Co.
Present position: Mine Sampler, Old Dominion Copper Mine.

Clair Lincoln Baker, El Paso, Tex.
Proposed by Kuno Doerr, H. A. Prosser, James Heggie.
Born 1880, Roxbury, Mass. 1898–1902, Tufts College, Chemical course, B. S. 1903–07, various positions with American Smelt. & Ref. Co.—Assayer, Chemist, apprentice in smelter, ore buyer. 1908–09, General Agent, Compania Minera de Penoles, Mapimi, Dgo., Mex. 1910–16, Ore buyer, Asst. Genl. Mgr. and later Genl. Mgr., American Smelt. & Ref. Co., Mexico.
Present position: Genl. Mgr., Mexican Dept., American Smelt. & Ref. Co.

George Pomeroy Bartholomew, New York, N. Y.
Proposed by Edwin S. Berry, Pope Yeatman, Fred Hellmann.
Born 1874, Pottsville, Pa. 1896, Grad., Lehigh Univ., Bachelor of Met. 1898–1900, Rand Mines, Ltd., Johannesburg, S. Africa. 1900–06, Jeamsville Iron Works Co., and Rochester & Pittsburgh Coal & Iron Co. 1907–09, Min. Engr., E. I. du Pont de Nemours Powder Co. 1909–12, Min. Engr., New York City.
Present position: 1912 to date, Asst. Cons. Engr., M. Guggenheim Sons, Chile Exploration Co., Braden Copper Co.

Charles J. Brown, Donora, Pa.
Proposed by C. L. Miller, B. D. Quarrie, H. A. Barren.
Born 1883, Sherman, N. Y. 1902, Grad., Central High School, Cleveland, O. 1906, Case School of Applied Science, Chem. Engrg. 1906, Union Carbide Co., Sault Ste. Marie, Mich. 1907, American Carbolite Co., Duluth, Minn. 1908, Zenith Furnace Co., Duluth, Minn. 1909, American Steel & Wire Co., Blast Furnace Dept., Cleveland, O.
Present position: Supt., Blast Furnaces, American Steel & Wire Co.

Samuel Rollins Brown, Plymouth, Cal.
Proposed by H. W. Hardinge, R. B. T. Kiliani, Ronald Clark.
Born 1888, Colona, Colo. 1902–06, Grad., Montrose County High School, Montrose, Colo. 1906–07, 1908–11, Colorado School of Mines, Golden, Colo., E. M. 1907–12, In mill Tomboy Gold Mines, Ltd., Telluride, Colo. 1912–13, Mineral Inspector, G. L. O. under F. C. Desendorf, Chief of Field Division. 1913, Prospecting in British Columbia. 1914, In mill, Tomboy Gold Mines, Ltd., Telluride, Colo.
Present position: 1914 to date; Mill Supt., Plymouth Cons. Gold Mines, Ltd.

Arthur Roy Campbell, Iola, Kans.
Proposed by Sidney J. Jennings, C. F. Moore, G. W. Metcalfe.
Born 1875, Belmont, Mass. 1896–99, Worcester Academy, Worcester, Mass. 1899–1903, Harvard Univ., B. S. 1903–04, El Oro Min. Co., Ltd., El Oro, Mex. 1904–13, U. S. Smelt. Co. Three years practical experience, three years head of different departments, two years, Asst. Supt., one year Supt. of plants, Lead and Copper Smelt. 1913, Asst. Mgr., American Zinc, Lead and Smelt. Co.
Present position: Zinc Smelter Supt., U. S. Smelt. Co.

James H. Causten, Lovelock, Nev.
Proposed by Francis Church Lincoln, J. C. Jones, Andrew W. Newberry.
Born 1868, Washington, D. C. Public Schools, Washington, D. C. Spencerian Business College, Washington D. C. Three and one half years, St. John Academy,

Alexandria, Va. (preparatory). One year, Civ. Engrg., Univ. of Virginia, Charlottesville, Va. 1915, Prospectors course, Univ. of Nevada. 1891–93, City Surveyor, Port Townsend, Wash. 1894–1908, U. S. Customs Service, Washington State, Alaska, Porto Rico, the last five years Collector of Customs for Porto Rico and for short period before going to P. R. a special Agent of U. S. Treasury. Since 1908, Mine operator in Nevada with one year as Min. Director Eureka Winsfall Mining Co.
Present position: Mine Operator, Antelope Springs, Nev.

Alvin M. Cummings, Mineville, Essex Co., N. Y.
Proposed by H. Comstock, George C. Foote, F. S. Witherbee.
Born 1886, Lyon Mountain, N. Y. 1909–12, Michigan College of Mines, Houghton, Mich. 1907–09, Engrg. Dept., Witherbee, Sherman & Co.
Present position: 1912 to date, Asst. Mine Supt., Witherbee, Sherman & Co.

Marcel (Robert Dyonis) Daly, Seattle, Wash.
Proposed by Amos Slater, Charles E. Weaver, Milnor Roberts.
Born 1860, Wissous, near Paris, France. 1878, Univ. of Paris, B. A. and B. S. 1883, Grad. as Engr., Ecole Centrale des Arts et Manufactures. 1889, Grad., School of Law, Paris. 1883–84, in the survey office, Northern French Ry. 1885–95, worked with father, Cesar Daly, architect of the French Govt. (survey office, constructions, technical publications) and travelled extensively in Europe and N. Africa. 1896–97, Appointed by the Societe Nationale de Credit, Paris, for examining manganese deposits in Southern Russia and the Caucasus. Visited and studied the principal oil fields of S. E. and Central Europe. 1897–1901, Chief Engr., Societe de Construction de Willebrook, Belgium near Brussels (steel works). 1901–05, Genl. Mgr., Societie An. des Forges d'Aiseau, near Charleroi, Belgium, (steel works and general machine construction). During the stay at Willebrook and Aiseau, have designed and erected numerous bridges, metallic constructions, machinery for quarries, etc. Worked for the City of Brussels, the Govt. of Belgium, the Free State of the Congo, the ports of Barcelona (Spain), Constanza (Rumania) Ostend (Belgium). 1905–10, Genl. Mgr., and Attorney in fact, Cia. Francaise de Petroles (French Petroleum Co.) in U. S.
Present position: 1911 to date; Cons. Engr. and Oil man.

Alexander Deussen, Houston, Tex.
Proposed by E. F. Burchard, David White, E. T. Dumble.
Born 1882, San Antonio, Tex. 1903, Univ. of Texas, B. S. 1904, Univ. of Texas, M. S. 1907–09, Geologic Aid, U. S. Geological Survey. 1909–15, Asst. Geol., U. S. Geological Survey.
Present position: Geol., Gulf Production Co.

Arthur Grant Ellis, Hayden, Ariz.
Proposed by David D. Moffat, Louis S. Cates, J. J. Ormsbee.
Born 1876, Boulder, Colo. 1891, East Denver High School. 1891–1903, Assayer, Chemist, Ore Testing, with Henry E. Wood, Denver. 1904–05, Mill Supt., Hanna Min. & Mill. Co., Lake City, Colo. 1905–08, Supt., Zinc Mills, Lanyar Zinc Co., Iola, Kans. 1908 to date, General Mill Foreman, Jackling Interests, Utah & Ariz.
Present position: Genl. Mill Foreman, Ray Cons. Copper Co.

Clarence T. Emrich, Globe, Ariz.
Proposed by W. B. Cramer, J. S. Low, L. O. Howard.
Born 1883, New York. 1902–04, Colorado College. 1905–09, Colorado School of Mines, E. M. 1909–11, Chief Chemist, North American Smelter & Mines Co. 1911, Chemist and Assayer, Pioneer Smelt. Co. 1912–13, Smelter Supt., Santa Fe Gold & Copper Min. Co. 1914, Professional work, Denver. 1915, Engr., Cariman Min. & Mill. Co.
Present position: Chemist, Old Dominion Copper Min. & Smelt. Co.

Harold Bertels Fell, Wilkes-Barre, Pa.
Proposed by John Lloyd, Paul Sterling, J. M. Humphrey.
Born 1889, Wilkes-Barre, Pa. 1895–1901, Public Schools, Wilkes-Barre, Pa. 1901–06, Harry Hillman Academy, Wilkes-Barre, Pa. 1906–07, Mercersburg Academy, Mercersburg, Pa., graduated in English Scientific course. 1907–12, Princeton Univ., C. E., Princeton, N. J. 1909–10, summers, Mechanical Engrg. Dept., Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1911, summer, 1912–13, Efficiency Engrg. Dept., Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1913–16, Wyoming Division, Engrg. Dept.
Present position: Supt. and Engr., Wyoming Valley Water Supply Co.

Jorge Belmiro de Araujo Ferraz, Rio de Janeiro, Brazil, S. Amer.
Proposed by L. C. Graton, Horace E. Williams, W. Lindgren.
Born 1883, Rio de Janeiro. Grad., Polytechnical School of Rio de Janeiro. 1908–14, Asst. Prof. of Physics and Chemistry, Pedagoquim Municipal, Rio de Janeiro. Life member of the Italian Geol. Society. Secy. and Treas., Brazilian commission of the International Exhibition of Turin-Rouss 1911. Knight of the order "Crown of Italy." Honorary member of the National Society of Agriculture.
Present position: Asst. Geol. and Petrographer, Servico, Geologico and Mineralogico do Brazil.

Dexter Bemis Followill, Collinsville, Ill.
Proposed by W. E. Newnam, J. A. Caselton, Herman Garlichs.
Born 1882, Rolla, Mo. Public and High School, Rolla, Mo. 1902–06, Missouri School of Mines. 1906–09, Chemist, Amer. Smelt. & Ref. Co., Leadville, Colo. 1909–12, Chemist, St. Louis Smelt. & Ref. Co., Collinsville, Ill. 1912–14, Chemist, Amer. Smelt. & Ref. Co., Perth Amboy, N. J. 1914–15, Chemist, St. Louis Smelt. & Ref. Co., Collinsville, Ill.
Present position: 1915 to date; Supt. of Basic Lead Sulphate Plant, St. Louis Smelt. & Ref. Co.

Sutter Albert Gardanier, Douglas, Ariz.
Proposed by Theodore Knutzen, Courtenay DeKalb, H. O. Hammond, J. M. Bonsall.
Born 1881, Cleveland, O. 1899, Two years East Denver High School. 1900–04, Asst. Assayer, Wheeler & Wheeler, Creede, Colo. 1905, Assayer, Creede United Mines Co., Creede, Colo. 1907, Mgr., Wheeler & Wheeler Assay office and bought office the same year. Sold the office 1910. 1912, Mgr., Las Tepalcates Mines, Sinoquipe, Son., Mex. 1912, bought assay office, Ben Norton, Douglas, Ariz. and have been here ever since.
Present position: Mgr., Douglas Assay Co.

Charles E. Griswold, Globe, Ariz.
Proposed by W. B. Cramer, L. O. Howard, I. H. Barkdoll.
Born 1874, Meriden, Conn. 1893–94, Cornell Univ. 1906–07, Chemist, San Fernando Mine, Lower Cal. 1913, Experimental work, at Smelter, Old Dominion Copper Min. & Smelt. Co. 1914–16, Experimental work, at Concentrator, Old Dominion Copper Min. & Smelt. Co.
Present position: Mill work, Old Dominion Copper Min. & Smelt. Co.

Justice Grugan, Edwards, St. Lawrence Co., N. Y.
Proposed by Cecil Pocock, W. D. B. Motter, Jr., William Campbell.
Born 1878, Philadelphia, Pa. Common Schools, Philadelphia. 1895–97, Drexel Institute, Philadelphia. 1903–04, private work with Prof. Amos P. Brown, Univ. of Pa., mineralogy. 1897–1902, Mineralogist, Geol., & Chem., Foote Mineral Co., Philadelphia, Pa. 1905–07, Travelling representative in Europe and mining rare ores in Australia, Foote Mineral Co. 1907–12, mine examinations, also general practice, Worth Bros. Co., Coatesville, Pa. 1912–16, Northern Ore Co., Edwards, N. Y.
Present position: Mgr., Northern Ore Co.

Horace S. Gulick, St. Louis, Mo.
Proposed by J. B. Emerson, J. D. Robertson, Herman Garlichs.
Born 1885, Orlando, Fla. 1903, A. & M. College, B. S., Okla. 1903–04, Asst. to Div. Engr., C. R. I. & P. Ry. Co., Oklahoma City, Okla. 1904–05, Mining, Milling & Developing Co., Zinc, Ark. 1905–13, Asst. Chem. & Chief Chem., American Steel Foundries, East St. Louis, Ill. 1913 to date; More-Jones Brass & Metal Co., St. Louis, Mo.
Present position: Met. Engr., More-Jones Brass & Metal Co.

Raymond Guyer, Ketchum, Ida.
Proposed by Charles V. Drew, H. S. Emlaw, Edmond N. Skinner.
Born 1880, Helena, Mont. 1895–98, Bordentown Military Institute, N. J. 1900–03, Columbia School of Mines. 1899, Backus & Johnston Co., Casapalca, Peru, S. Amer. 1903, Laborer and general utility man, De Lamar Plant, Carteret, N. J. 1903, Frank Klepetko's Office, N. Y. 1903–05, Supt., Backus & Johnston Co., Casapalca, Peru. 1905–06, L. A. Proano, Morococha, Peru, S. Amer. 1907–08, L. A. Proano, Tamboraque Smelter, Mining. 1908–09, Frank Klepetko, Lima Office,

Peru. 1909–11, Genl. Mgr. & Cons. Engr., E. Ferandiru, Cerro de Pasco, Peru. 1911–12, W. R. Grace & Co., Lima, Peru, S. Amer. 1913–16, General mining work in U. S. and Peru, S. Amer.
Present position: Reporting and examination.

Bernard Hasbronck, Tulsa, Okla.
Proposed by Dorsey Hager, R. A. Conkling, H. A. Wheeler.
Born 1882, Hackensack, N. J. 1907, Princeton Univ., B. S. in Geol. 1908–11, Special in Min. Engrg., McGill Univ. 1912–13, Field Geol., Caribbean Petroleum Co., Caracas, Venezuela. 1913–14, Geol. in charge Southern Oil Fields, Compania Mexicana de Petroles "El Aguila," S. Amer., Tampico, Mex. 1914, Genl. Mgr., Southern Oil Fields, Cia. Mexicana de Petroles "El Aguila," S. Amer.

Albert A. Heimrod, New York, N. Y.
Proposed by Herman Garlichs, L. V. Emanuel, H. H. Utley, R. G. Hall.
Born 1885, Hanover, Germany. 1902, Grad., Omaha High School. 1906, Grad., Univ. of Nebraska, B. S. 1910, Columbia Univ., Met. Dept., Master of Arts. 1906–08, General Engrg. work, Chicago & Burlington & Quincy R. R. 1910–11, General foreman, Gold and Silver Dept., Balbach S. & R. Co., Newark, N. J. 1911–12, Foreman tank room, anode furnaces, and blue vitriol copper dept., Balbach S. & R. Co. 1910, On retort, cupel and desilverizing work, Balbach S. & R. Co. 1913–15, Constr. Engr., Balbach S. & R. Co., New Lead plant, Newark Bay Front.
Present position: Engr., Research Corpn.

W. Homer Hendricks, New York, N. Y.
Proposed by William H. Shearman, Thomas T. Read, George C. Stone.
Born 1878, Hummelstown, Pa. 1906, Met. Engr., Lehigh University. 1906, Asst. Supt., Acid and Lithopone Dept., N. J. Zinc Co. (of Pa.) Handled one or both of these departments since 1908.
Present position: Genl. Sales Engr., N. J. Zinc Co.

William Henry Keefe, Jr., Globe, Ariz.
Proposed by W. B. Cramer, G. N. Bjorge, I. H. Barkdoll.
Born 1889, Derby, Conn. 1911, Grad., Sheffield Scientific School, Yale Univ., Ph. B. 1911–13, Miner, Penn Iron Min. Co. 1914, Asst. Engr., Verona Min. Co. 1915–16, Assayer and Chemist, United Verde Copper Co.
Present position: Asst. Engr., Old Dominion Copper Min. & Smelt. Co.

John Charles Kinnear, McGill, Nev.
Proposed by R. E. H. Pomeroy, A. G. Marsh, C. B. Lakenan.
Born 1885, Carnonstie, Scotland. 1890–1902, Common and High School, Salem, Mass. 1902–07, Mass. Inst. of Tech., S. B. 1906, Board of Water Supply City of New York. 1907, Goldfield Cons. Mines Co., Goldfield, Nev. 1908–10, Goldfield Daisy Min. Syndicate, Goldfield, Nev. 1910, Bingham and Garfield Ry. Co., Bingham, Utah. 1910–16, Nevada Cons. Copper Co., Steptoe Plant, McGill, Nev.
Present position: Asst. Smelter Supt., Steptoe Plant, Nevada Cons. Copper Co.

Alfred George Kirby, Toronto, Ont., Canada.
Proposed by H. W. Hardinge, C. W. Merrill, Charles A. Chase.
Born 1871, Oxford, England. No technical education other than self acquired. 1892–96, Providence Gold Mining, Nevada City, Cal. 1896–98, Leasing Smuggler-Union Co., Telluride, Colo. 1898–1901, General Mill Construction, Mount Wilson G. & S. Min. Co., Mt. Wilson, Colo. Golden Fleece Min. Co., Lake City, Colo., etc. 1901–06, Mill Supt. and general construction, Liberty Bell G. & S. Min. Co., Telluride, Colo. With the same company at Combination Mill, Goldfield, Nev. Upon transfer of property to Goldfield Cons. Ltd., with this property till end of 1908. Designed and erected the Nova Scotia Mill, Cobalt, Ont. 1909, Remained with company as a Manager till 1912. During 1911 and 1912, Cons. Engr. for the Hollinger Gold Mines & McIntyre Mines, Porcupine, Ont. Designed and erected the first mills for these companies. 1912–16, Genl. Mgr., Dominion Reduction Co., Cobalt, Ont.
Present position: Consulting Practice.

William Hoffman Kobbe, St. Louis, Mo.
Proposed by Ralph Arnold, Irving Perrine, J. William Smith.
Born 1881, Ft. Hamilton, N. Y. 1900, Grad., Lowell High School, San Francisco, Cal. 1901, course, Biltmore Forest School, Biltmore, N. C. 1901, Member, California Academy of Sciences. 1904, Grad., Yale Forest School, Yale Univ. Since then

close student of Petroleum and patentee of devices used in the oil industry. 1904–07, Government forester in Philippine Islands and Arizona. 1908, Foreman, Globe Exploration Co., Cal. 1910, Supt. above Co. and also of Rock Oil Co. 1913, Div. Supt., General Petroleum Co. 1914, Asst. Field Mgr., Standard Oil Co. in Turkey until outbreak war.
Present position: Petroleum Engr., Pierce Oil Corpn.

Carl A. Lemke, Midvale, Utah.
Proposed by Ernest Gayford, John M. Callow, H. H. Utley.
Born 1880, Brooklyn, N. Y. 1896, High School. 1900, Colorado School of Mines, E. M. 1900, Assayer, Copper Hill Min. Co., Rinconada, N. Mex. 1900–01, Chemist, Silver Lake Mines, Silverton, Colo. 1902–04, Assayer and Surveyor, Uncle Sam Mine, Swandyke, Colo. 1905, Land surveying, Fresno, Cal. 1906–12, Mill Foreman and Supt., American Zinc Extraction Co., Leadville, Colo.
Present position: 1912 to date; Asst. Supt. in charge of Concentrator, U. S. Smelt. Co.

Edward Dale Lynton, San Jose, Cal.
Proposed by George Kingdon, M. J. Elsing, Royal P. Jarvis.
Born 1890, New York City. 1900–04, Preparatory School, Reigate, England. 1904–08, Harrow School, Harrow, England. 1908–12, Missouri School of Mines, Rolla, Mo., B. S. 1912–15, Practical experience and thesis received, E. M. 1912–14, Geol. Dept., Cananea Cons. Copper Co. 1914–15, churn drills, Detroit Copper Min. Co., Morenci, Ariz.
Present position: 1915 to date; Chief Engr., Democrata Min. Co.

William Arthur McCoy, McGill, Nev.
Proposed by R. E. H. Pomeroy, A. G. Marsh, C. B. Lakenan.
Born 1885, Riverton, Va. 1908, Washington and Lee Univ., B. S. 1908–09, Chem., iron mine, for H. J. Seibel, Philadelphia, Pa. 1909–13, Chem., New Jersey Zinc Co., Franklin, N. J. 1913–14, Chem., American Smelt. & Ref. Co., Maurer, N. J. 1914, Chem. and in charge of experimental work, Chile Copper Co., Amer. Smelt. & Ref. Research Laboratory, Maurer, N. J. 1915, Chief Chem., for Dr. John H. Banks, New York, N. Y.
Present position: Chief Chem., Nevada Cons. Copper Co.

Austin Lee McRae, Rolla, Mo.
Proposed by H. A. Buehler, Charles Y. Clayton, Robert J. Anderson.
Born 1861, McRae, Ga. 1881, Univ. of Georgia, B. S. 1886, Harvard Univ., S. D. 1889–91, Asst. Prof. Physics, Univ. of Missouri. 1891–94, Prof. of Physics, Missouri School of Mines. 1894–1906, Associate Prof. Physics, Univ. of Texas. 1896–99, Cons. Engr., St. Louis, Mo. 1899–1915, Prof. Physics, Missouri School of Mines.
Present position: 1915 to date; Director and Prof. Physics, Missouri School of Mines.

Hugh Renwick MacMichael, El Paso, Tex.
Proposed by Kuno Doerr, H. A. Prosser, James Heggie.
Born 1881, Springfield, Ill. Education entirely by private study. 1900—03, Asst. to Supt., construction and later in charge rolling mill dept. of engineering office, Colorado Fuel & Iron Co., Pueblo, Colo. 1904–06, Engr. in charge design and construction of smelter, American Smelters Securities Co., Asarco, Dgo., Mex. 1906–07, in charge of construction of smelter, American Smelt. & Ref. Co., Chih., Mex.
Present position: 1907 to date; Chief Engr., American Smelt. & Ref. Co. Southern Dept.

Jacob A. Martin, Santa Rita, Grant Co., N. M.
Proposed by William J. Hallett, W. G. Haldane, F. W. Traphagen.
Born 1875, Freukendorf, Switzerland. 1907, Colorado School of Mines, E. M. 1907, Asst. Engr., Minas Tecolotes y Annexas, Santa Barbara, Chih., Mex. 1907–08, Miner, American Eagles, Cripple Creek, Colo. 1908, Timberman, Portland Gold Min. Co., Victor, Colo. 1908–09, Millman, American Zinc Extraction Co., Leadville, Colo. 1909–10, Miner, Fanny Rawlins, Leadville, Colo. 1910–12, Asst. Engr., Superior Coal Co., Superior, Wyo. 1912–14, Contracting, Superior Coal Co., Superior, Wyo. 1914, Foreman, Superior Coal Co., Superior, Wyo. 1914, Foreman, Union Pacific Coal Co., Rocksprings, Wyo. 1914–16, Contracting, Union Pacific Coal Co., Hanna, Wyo.
Present position: Asst. Engr., Chino Copper Co.

Richard W. Mayne, Globe, Ariz.
Proposed by W. B. Cramer, I. H. Barkdoll, G. N. Bjorge.
Born 1869, Nebraska. 1891–96, Mining, Cal. 1896–1900, La Fortuna, Ariz. 1900–16, Old Dominion Copper Min. & Smelt. Co.
Present position: Mine Foreman, Old Dominion Copper Min. & Smelt. Co.

Cyrus Edward Minor, Carrizozo, N. Mex.
Proposed by J. N. Houser, M. H. Newman, A. W. Koch.
Born 1882, Bellville, Ill. 1904, Grad., Missouri School of Mines, B. S. in Min. Engrg. 1904, Clark Engrg. Co., Bisbee, Ariz. 1904–05, Surveying, Calumet & Arizona Copper Co., Bisbee. 1905–07, Asst. Engr., Cananea Cons. Copper Co., Cananea, Son., Mex. 1907–10, Member firm "Minor and Hart," Cananea, Son., Mex. 1910–13, Asst. Engr., for above-mentioned Cananea Cons. Copper Co. 1913–15, Asst. Engr., Chicago and Western Indiana R. R., Chicago, Ill. 1916, Min. Engr., on exploration work for eastern capital.
Present position: Min. Engr. for parties in the East.

Jones Guerry Moore, Birmingham, Ala.
Proposed by J. C. Maben, Jr., E. Francis McCrossin, James Bonnyman.
Born 1857, near Tuskegee, Ala. Education in Common School, Alabama. Coal Mining for thirty years, Milner Coal & R. R. Co., Tenn. Coal & R. R. Co.
Present position: Genl. Supt. of coal mines, Sloss Sheffield Steel & Co.

Harry Cooper Parker, McGill, Nev.
Proposed by R. E. H. Pomeroy, A. G. Marsh, C. B. Lakenan.
Born 1881, Wellsville, Utah. 1897–1900, High School Dept., Utah Agricultural College, Logan, Utah. 1900–01, Freshman, Utah Agricultural College. 1901–04, Student, Harvard Univ., Lawrence Scientific School, S. B. 1904–05, Construction work, Newhouse Mines and Smelter Corpn. 1905–06, Highland Boy Smelter, Utah Cons. Min. Co. 1906–08, Engrg. business, Salt Lake City, Utah. 1908–10, Teaching, Prof. Geol. and Mineralogy, Utah Agricultral College, Logan, Utah. 1910–11, Mgr., Nev. Utah Mines and Smelters, Corpn., Pioche, Nev. 1912 to date, Met. Engr., Roasting Dept., Nevada Cons. Copper Co.
Present position: Smelter Engr., Nevada Cons. Copper Co.

Enoch Perkins, Juneau, Alaska.
Proposed by O. P. Rogers, B. L. Thane, E. V. Daveler.
Born 1891, Pendleton, Ore. 1897–1905, Boise Public Schools, Boise, Idaho. 1905–09, Grad., Boise High School, Boise, Idaho. 1909–13, Grad., Univ. of Idaho, in Mining and Civ. Engrg., B. S. (C. E.). 1912–13, Deputy County Engr., Latah Co., Idaho. Deputy U. S. Mineral Surveyor for Idaho (still hold commission). Licensed Land Surveyor for Idaho. 1914, Surveyor, Alaska Gastineau Min. Co., Juneau, Alaska. Wulzen Alaska Gastineau Min. Co. 1914–15, Engr., Ebner Mine, U. S. Smelt. Ref. and Min. Exploration Co.
Present position: 1915 to date; Surveyor and Draughtsman, Alaska Gastineau Min. Co.

William Bemiso Phelps, Knik, Alaska.
Proposed by Amos Slater, I. F. Laucks, James A. Kelly.
Born 1883, New Orleans, La. 1907, Colorado School of Mines, E. M. 1907–09, Sampler, Surveyor, Shift-Boss, Surface Foreman, Ray Cons. Copper Co., Ray, Ariz. 1909–10, Mine Foreman, Goldroads Min. Co.. Goldroads, Ariz. 1910–12, Examination work in Mexico and later in charge of Casados Mine in Jalisco. 1912–13, Shift-Boss and later Foreman, Goldroads Min. Co. 1913–16, At Casados Mine as Supt. and in Alaska as Mill Supt. for Alaska Free Gold Min. Co.
Present position: On examination work and charge of exploration work in Alaska for Dr. H. O. Schaleben, Seward, Alaska.

Rudolph Porter, Federal, Ill.
Proposed by W. E. Newnam, William Allen Smith, Philip N. Moore.
Born 1873, Cincinnati, O. Public Schools, Cincinnati. 1888–1900, Pueblo Plant, Amer. Smelt. & Ref. Co., Pueblo, Colo. 1900–16, Federal Lead Co., Federal, Ill.
Present position: Mgr., Federal Lead Co.

Guy M. Powell, Kellerman, Ala.
Proposed by Warren L. Kluttz, C. T. Fairbairn, Walter J. Pennhallegon.
Born 1879, Smethport, Pa. 1896, High School, Ridgway, Pa. 1896–99, Appren-

tice and draftsman, Ridgway Dynamo and Engine Co., Ridgway, Pa. 1901 Draftsman, Julian Kennedy, Pittsburgh, Pa. 1902–03, Draftsman, S. V. Huber & Co., Pittsburgh, Pa. 1903–05, Draftsman, Riter Conley Mfg. Co., Pittsburgh, Pa. 1905–07, Draftsman, Construction Engr., Illinois Steel Co., South Chicago, Ill. 1907–09, Construction Engr., Tennessee Coal, Iron & R. R. Co., Ensley, Ala. 1910–11, Construction Engr., various contract jobs. 1912 to date, Engr., Central Iron and Coal Co., Hott, Ala.
Present position: Supt. of Coal Mines.

George C. Riser, Jr., McGill, Nev.
Proposed by R. E. H. Pomeroy, A. G. Marsh, C. B. Lakenan.
Born 1876, Salt Lake City, Utah. 1891, District School, Salt Lake City, Utah. 1896, Univ. of Utah Preparatory School, Salt Lake City, Utah. 1904, Univ. of Utah, B. A. 1904–05, Instructor, Univ. of Utah. 1896–1900, Worked in the following mines and mills, Operator in mill and refinery, Delamar Mill, Delamar, Nev. Mucker, Timberman, Helper and Miner, Dailey West & Silver King Mines, Park City, Utah. Repairman and Operator, Golden Gate Mill, Mercur, Utah. Timberman, Miner, Repairman in mill, Anny Lary Min. Co., Kimberly, Utah. 1903–04, Summer, Copperton Mill, Utah Copper Co. 1905–06, Experimental work, Copperton Mill, Utah Copper Co., Bingham Canyon, Utah. 1906–08, Erector, Demonstrator and Salesman, Joshua Handy Iron Works, San Francisco, Cal. 1908, Salesman, Fairbanks, Morse & Co., Salt Lake City, Utah. 1906–15, Experimental and construction work, Concentrating Dept., Shift and General Mill Foreman, Utah Copper Co. During this period I was sent to the various mills controlled by D. C. Jackling to do experimental work and assist in starting up. Started up Ray Cons. Mill, Hayden, Ariz. and acted as Supt. of Mill.
Present position: 1915 to date; Supt., Concentrator, Nevada Cons. Copper Co.

T. C. Roberts, Clarkdale, Ariz.
Proposed by Will P. Clark, H. N. Thomson, A. T. Coston.
Born 1876, New Orleans, La. Tulane Univ., Electrical and Mechanical courses, New Orleans. Chief Engr., general engineering work, American Smelt. & Ref. Co. for several years. For five years, Arkansas Valley Railway Light and Power Co., Pueblo, Colo. In all of the above positions the design and construction of all engineering work has been under my immediate supervision, including power plants, smelters, utility and improvement companies, electrical and steam railroads, various types and lengths of high tension transmission lines, water power development, etc., also at various times I have constructed residences, school, business and other public buildings.
Present position: Chief Engr. of all Senator Clark's interests in Arizona.

Clyde Polhemus Ross, Globe, Ariz.
Proposed by W. B. Cramer, G. N. Bjorge, I. H. Barkdoll.
Born 1891, New York, N. Y. 1914, Mass. Inst. of Tech., S. B. 1915, Mass. Inst. of Tech. & Harvard, S. M. 1913, Ray Cons. Min. Co. 1915, Asst. to Prof. H. O. Hofman, Mass. Inst. of Tech. 1916, Asst. to Prof. Graton, Harvard.
Present position: Asst. Geol., Old Dominion Copper Min. & Smelt. Co.

Ray P. Saffold, Globe, Ariz.
Proposed by W. B. Cramer, L. O. Howard, I. H. Barkdoll.
Born 1878, Fountain Mills, Md. 1901–05, Lehigh Univ. 1905–06, Mill Foreman, Bamberger Delamar Mines, Delamar, Nev. 1906, Engrg., Green Cananea, Mex. 1907, Supt., Cananea Nueve, Mex. 1908–13, General Engrg. and professional work in Mexico. 1914–15, Experimental work (leaching) with Calumet & Arizona Min. Co. 1916, Private experimental work.
Present position: Private experimental work.

Howard P. Saunders, Herculaneum, Mo.
Proposed by Herman Garlichs, William Allen Smith, Franz Cazin.
Born 1865, Milan, O. 1884, Huron Institute. 1888, Ohio State Univ. 1893, Pueblo Smelt. & Ref. Co. 1898–1902, Mill and Smelter designing. 1902–03, Chief Engr., Old Dominion Copper Min. Co. 1904–05, Murray Plant, Amer. Smelt. & Ref. Co. 1905, U. S. S. R. & M. Co. 1906–07, Tintic Smelt. Co.
Present position: Chief Engr., St. Joseph Lead Co.

Guy Emerson Sheridan, Butte, Mont.
Proposed by W. C. Siderfin, J. F. Welborn, W. N. Rossberg.
Born 1879, Detroit, Mich. 1902, Univ. of Montana, B. S. 1902, Chem., Colorado

Smelt. & Min. Co. 1904, Chem., Colusa Parrot Min. & Smelt. Co. 1914, Chem., Timber Butte Mill. Co.
Present position: Chief Chem., Timber Butte Mill. Co.

Blake William Smettem, Newcastle, Tex.
Proposed by F. K. Copeland, E. J. Rossbach, D. F. Nance.
Born 1884, Kansas City, Mo. 1896–1900, Grammar School, Springfield, Mo. 1900–02, High School, Springfield, Mo. 1902–04, Private Tutor, Ft. Worth, Tex. 1906, Private Tutor, New Orleans, La. 1902–04, Rodman, Gulf Colorado & Santa Fe Ry., Texas. 1905, Rodman, St. Louis & San Francisco Ry., Memphis. 1905–06, Leveler, Louisiana & N. West. Ry., Homer, La. 1906–07, Inspecting Engr., The Rail Joint Co., New Orleans, La. 1908–10, Special Work Foreman, New Orleans Ry. & Lt. Co., New Orleans, La. 1911, Chief Engr., Stone & Webster, Eng. Corp., Ft. Worth, Tex. 1912–13, Chief Engr., Texas Building Co., Ft. Worth, Tex. 1913, Chief Engr., Tarrant Const. Co., Ft. Worth, Tex. 1914–15, Chief Engr., Strawn Coal Co., Strawn, Tex. 1915–16, Chief Engr., Belknap Coal Co., Newcastle, Tex.
Present position: Chief Engr., Belknap Coal Co.

Frank Norton Spencer, New York, N. Y.
Proposed by J. H. Janeway, J. E. Hayes, Jr., George C. Stone.
Born 1883, Toledo, O. 1901, Lawrenceville School. 1905, Princeton, Civ. Engrg. 1905–06, Engrg. Dept., Penna. Lines. 1906–07, Engr., Dittoe & Wisenalle, Architects, Cincinnati. 1907–11, Chief Engr., A Bentley & Sons Co., Toledo, O. 1912–16, Mgr., Leonard Const. Co., New York City.
Present position: Asst. to the Genl. Mgr., The New Jersey Zinc Co.

Leighton Stewart, New York, N. Y.
Proposed by Edmond N. Skinner, W. H. Aldridge, Lucius W. Mayer.
Born 1887, Summerside, Canada. 1909, Grad., McGill Univ., B. Sc. in Min. Engrg., Montreal, Canada. 1909, Engrg. and Sampling Dept., Ray Cons. Copper Co., Ray, Ariz. 1910–13, Min. Engr., at the Asientes, Charcas and Santa Barbara Units, Min. Dept., American Smelt. & Ref. Co., Mexico. 1913, Asst. Supt., Cusi Min. Co., Cusihuiriachic, Chih., Mexico. 1913–16, Field Engr., Andes Exploration Co., Santiago, Chile, S. Amer.

Kurt Stock, Bartlesville, Okla.
Proposed by K. C. Fraser, F. R. Bell, Harold C. Price.
Born 1874, Berlin, Germany. 1881–84, The complete type of education usual in Germany Schools (Real-Gymnasium) ending with the Abiturium. 1901–05, Studied at Royal Saxon School of Mines, Freiberg, Saxony, and obtained the degree of "Diplom-Huetten-Ingenieur" (Met. Engr.). 1894–1901, in different banking firms in Berlin, Germany and London, England. 1906, Asst. Chemist, Globe Smelter, Amer. Smelt. & Ref. Co., Denver, Colo. 1906–10, Chief Chemist and Asst. Supt., U. S. Zinc Co., Pueblo, Colo. 1910–13, Supt., Lanyon-Starr Smelt. Co. (The American Metal Co., Ltd.), Bartlesville, Okla. 1913–14, Supt. and Vice-Pres., Altoona Zinc Smelt. Co. (The American Metal Co., Ltd.), Altoona, Kans. 1914, Visit in Europe. 1914–15, Constructed the Clarksburg Clinkering Co.'s plant, Bridgeport, W. Va., for the American Metal Co., Ltd.
Present position: 1915 to date; Supt., Bartlesville Zinc Co. (The American Metal Co., Ltd.).

Sidney Marvin Stone, Virginia City, Nev.
Proposed by Charles Butters, Whitman Symmes, Francis Church Lincoln.
Born 1876, San Francisco, Cal. Grammar Schools, San Francisco, Cal. 1892, Grad., Boys High School. 1893–98, Purchasing agent, Sunmir and Jack Gold Min. Co., Germiston, S. African Republic. 1899–1900, Miner and Shift Boss, La Fortuna Mine, Blaisdell, Ariz. 1901–04, Employed in preliminary surveys of Western Pacific Railway. 1904–16, Mgr. for Chas. Butters, Kennett, Shasta Co., Cal. and Chas. Butters & Co., Ltd., Virginia City, Nev.
Present position: Mgr., Chas. Butters & Co., Ltd.

Roy Lesslie Wahl, Crosby, Minn.
Proposed by W. Wearne, F. W. McNair, F. W. Sperr.
Born 1891, Arnheim, Mich. 1909, Hancock High School, Hancock, Mich. 1912, Michigan College of Mines, Houghton, Mich., E. M. and B. S. 1912–15, Min. Engr., Rogers Brown Ore Co., Chicago, Ill.
Present position: 1915 to date; Min. Engr., Inland Steel Co.

Leland Edward Wemple, Hillsboro, Ill.
Proposed by Henry A. Wentworth, Charles E. Locke, Carle R. Hayward.
Born 1886, Waverly, Ill. 1902–04, Illinois College. 1904–08, Mass. Inst. of Tech., S. B. 1909–13, Supt., Smelting Dept., Hoyt Metal Co., St. Louis, Mo. 1913–14, Asst. Mgr. Smelters, Caney, Kans. 1914, Met., American Zinc Lead & Smelt. Co., Hillsboro, Ill.
Present position: Met., American Zinc Lead & Smelt. Co.

Leslie Vickery Waterhouse, Queenstown, Tasmania.
Proposed by Robert P. Roberts, Robert Sticht, Charles E. Cook.
Born 1886, Sydney, N. S. W. 1898–1904, Sydney Grammar School, N. S. W. 1905–09, Sydney Univ., N. S. W., B. E. 1910, Member, Australasian Institute of Min. Engrg. 1914, Associate Member, Institute of Min. & Met. 1910, Assayer Broken Hill Block 14 Silver Min. Co. 1910–12, Experimental Met., Broken Hill Junction North Silver Min. Co. 1913 to date, Flotation Experimentalist, and Mill Supt., The Mount Lyell Min. & Ry. Co., Ltd.
Present position: Mill Supt., The Mount Lyell Min. & Ry. Co., Ltd.

Associate Members

Edward F. Boyle, St. Louis, Mo.
Proposed by J. D. Robertson, John B. Emerson, H. A. Wheeler.
Born 1884, Brooklyn, N. Y. Brooklyn, N. Y. Public Schools, and acquainted with faculty of Pratt Institute, also extensive instruction under H. T. Beckwith, Palo Alto, Cal. Leland-Stanford University also Mr. Barringer of Philadelphia. Very near fifteen years with Ingersoll-Rand Co. in general mining and compressor work. Also expert core-driller in West Va., New Mexico, Arizona and N. Y. State for Ingersoll-Rand Co.
Present position: Sales Engr., Ingersoll-Rand Co.

A. Palmer Coombe, Cleveland, O.
Proposed by R. P. Tinsley, George D. Barron, I. P. Lihme.
Born 1864, Bermuda.

James McClure Platt, Alamosa, Colo.
Proposed by W. A. Butchart, L. A. Delano, H. Rabling.
Born 1883, Elizabeth, N. J. 1893–98, Grammar and High School, Colorado Springs, Colo. 1898–99, East Denver High School. 1902–06, Colorado College, School of Engrg. 1911, Colorado College, B. S. 1899–1902, with J. R. Rockfellow, Tombstone, Ariz. 1907, Mine Surveyor, Dolores Mine, American Smelters Securities Co., Matehuala, S. L. P., Mex. 1908, Engr., Cia. Min. La Quimica y Anexas, Sultepec, Est. de Mex., Mex. 1909–16, Supt., Guadalupe Arcos Mine, Zacualpan, Est. de Mex., Mex.
Present position: In charge of mining operations of Mr. G. E. Trager.

Edward Taylor Wilson, Denver, Colo.
Proposed by R. P. Tinsley, Ernest R. Ackerman, Henry D. Hibbard.
Born 1869, Ballston Spa, N. Y. High School. Thirty years in the petroleum oil business, twenty-six of which have been with the Standard Oil Co.
Present position: Pres., The Continental Oil Co.

George Wingfield, Reno, Nev.
Proposed by E. A. Julian, J. W. Hutchinson, Francis Church Lincoln.
Born 1876, Fort Smith, Ark. Pres., Goldfield Cons. Mines Co., Nevada Hills Min. Co., Buckhorn Mines Co. and a number of others for several years.
Present position: Pres. of above companies.

Junior Members

James Andrew Bruce Armstrong, New York, N. Y.
Proposed by Charles P. Berkey, R. M. Raymond, Robert Peele.
Born 1894, New York City. 1900, Public School, New York City. 1908, Flushing High School, L. I. 1912, Surveyor for C. U. Powell, Flushing, L. I. 1915, Asst. to Master Mech., Lehigh Coal and Nav. Co., Lansford, Pa.
Present position: Senior, Columbia Univ.

Michiya Hiraoka, Boston, Mass.
Proposed by Charles E. Locke, Carle R. Hayward, Edward E. Bugbee.
Born 1877, Tokyo, Japan. Primary School. 1895, Middle School. 1901, High School. 1904, Tokyo Imperial Univ. 1904–08, Engr., Kasakura & Ashio Copper Mines. 1908–15, Teacher, Osaka Technical College.
Present position: Student, Mass. Inst. of Tech.

Harry Daniel Kline, Rolla, Mo.
Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.
Born 1892, Rolla, Mo. 1910, Grad., Rolla High School. 1910, Surveyors' helper, Utah Copper Co., Bingham, Utah. 1911–12, Mining and Sampler, Utah Copper Co., Bingham, Utah. 1913–14, Mining, Buckhorn Mines Co., Buckhorn, Nev. 1915, Mining, Joplin, Mo.
Present position: Student, Missouri School of Mines.

Hunyet Lee, Golden, Colo.
Proposed by William R. Chedsey, F. W. Traphagen, Harry J. Wolf.
Born 1892, Kuantung, China. Prepared for college in Tokyo, Japan.
Present position: Senior Student, Colorado School of Mines.

William Walinsley Lynch, New Haven, Conn.
Proposed by J. F. McClelland, L. W. Bahney, Arthur F. Taggart.
Born 1894, Rowayton, Conn. 1912–15, Sheffield Scientific School, Ph. B. 1915, Laborer, Canadian Copper Co., Creighton Mine, Ont.
Present position: Student, Sheffield Scientific School.

Carlos Aliaga Mariaca, La Paz, Bolivia, S. Amer.
Proposed by Franz Germann, J. F. Aguilar-Revoredo, F. J. Dorion.
Born 1882, La Paz. General education in the National College of La Paz. 1900–05, Technical education in the Escuela de Mineria at La Serena & Santiago, Chile. 1908, Engr., La Paz Min. Co. 1908–11, Compania Corocoro de Bolivia. 1912–15, Director, Escuela Nacional de Mineria, Oruro, Bolivia.
Present position: Consulting practice.

William Bennett Miller, Berkeley, Cal.
Proposed by Andrew C. Lawson, Charles E. van Barneveld, W. S. Morley.
Born 1888, Plattsmouth, Nebr. Up to 1909, Public Schools, San Bernardino. 1909–16, Univ. of Cal. 1913–14, Sub-level boss, Ray Cons. Copper Co., Shift-boss, Yellow Aster Min. & Mill. Co.
Present position: Student, Univ. of Cal.

Howard Leslie Minister, Canon City, Colo.
Proposed by Harry J. Wolf, F. W. Traphagen, E. P. Mathewson.
Born 1891, Zaleski, O. 1908, Grad., South Canon High School. Nine months, Diamond Fire Brick Co., Canon City, Colo. Seven months, U. S. Smelt Co., Canon City, Colo. Three and one-half years, Empire Zinc Co., Canon City, Colo. Four months, Wolf Park Coal Co., near Canon City, Colo.
Present position: Senior, Colorado School of Mines.

Will Merton Traver, Jr., Central City, Nebr.
Proposed by William Wraith, V. A. Hart, F. W. Traphagen.
Born 1894, Nampa, Ida.
Present position: Student, Colorado School of Mines.

Change of Status—Associate Member to Member

John William McBride, Spokane, Wash.
Proposed by L. K. Armstrong, J. McD. Porter, W. H. Linney.
Born 1871, Kilbourn, Wis.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period May 10, 1916 to June 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 114, February to June, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of June 10, 1916.

ADAMS, ARTHUR K.......................................High St., Spencer, Mass.
AGUILAR, PONCIANO..........................La Presa 87, Guanajuato, Mexico.
ALDERSON, VICTOR C., Mgr., Winnemucca Mining & Milling Co., Winnemucca, Nev.
ALLEN, C. A..3 Morse St., Dorchester, Mass.
ALLEN, MILTON A., Min. Engr., Arthur L. Pearse & Co.,
346 Broadway, New York, N. Y.
ALSDORF, FREDERICK C., Min. Engr......Great Northern Hotel, New York, N. Y.
ANDERSON, ROBERT J...............Amer. Rolling Mill Co., Middletown, Ohio.
ANDREEN, H. M...................................R. F. D. 1, Redwood City, Cal.
ARMSTEAD, DANIEL M....................300 No. Broadway, St. Louis, Mo.
BARBOUR, PERCY E., Managing Editor, Engineering and Mining Journal,
10th Ave. & 36th St., New York, N. Y.
BARTLETT, EDWARD E...................................Box 216, Sapulpa, Okla.
BASSETT, THOMAS E...........................Instructed to hold everything.
BENHAM, W. M......................Co. H, 1st Arizona Infantry, Douglas, Ariz.
BERGMAN, JULIUS GUSTAF...................1211 Mills Bldg., El Paso, Tex.
BIGELOW, BRAXTON, 170th Company Royal Engineers,
British Expeditionary Force, France.
BILHARZ, O. M., Min. Engr., Pres.......The O. M. Bilharz Min. Co., Miami, Okla.
BILLICK, DON C., Mill Supt..........................Comet Mine, Basin, Mont
BLAIR, ALBERT E., Genl. Mgr., Estates of Francisco Madero, University Club,
2a Bucareli 35, Mexico City, Mex.
BLAKE, T. W..South Haven, Mich.
BONSACK, ARTHUR A., Bonsack Machinery Co., 1935 Boatmen's Bank Bldg.,
St. Louis, Mo.
BORNHOFT, HENRY..........................P. O. Box 867, San Antonio, Tex.
BOYD, JESSE TAYLOR, Supt................Red Cliff Mines, Red Cliff, Colo.
BRADT, HARLAN H...Lake Mills, Wis.
BRINKER, ARTHUR C., Min. Engr., Mgr., The Buena Tierra Min. Co., Ltd.,
1209 Mills Bldg., El Paso, Tex.
BROCK, R. W..........Univ. of British Columbia, Vancouver, B. C., Canada.
BROOKE, LIONEL...................................Instructed to hold everything.
BROOKS, JOHN M., JR., Care Chile Exploration Co., Chuquicamata, Chile,
S. Amer., via Colon & Antofagasta.
BROMLY, A. H.......94 Lennard Road, Penge, London, S. E., England.
BROWN, ARTHUR H.......................133 Kent St., London, Ont., Canada.
BRYDEN, C. L.........U. S. Smelt. Co., Inc., 120 Broadway, New York, N. Y.
BURBRIDGE, THOMAS B...................................Box 660, Denver, Colo.
BURNS, J. J..........................Care Leonard Hotel, Butte, Mont.
BURRAGE, ROBERT H...Needham, Mass.
CAHEN, JAMES P., JR...............353 Central Park West, New York, N. Y.
CAMPBELL, W. C.............................1454 Logan St., Denver, Colo.
CARLSON, A. E................Care The Shannon Copper Co., Metcalf, Ariz.
CARPENTER, EDWIN LISSEL, Vice-Pres., Chesapeake and Ohio Coal and Coke Co.,
120 Broadway, New York, N. Y.
CARPENTER, JAMES R.......................1080 North Ave., Elizabeth, N. J.
CARSTENS, C. E..Ackley, Ia.
CHAPMAN, LEWIS C..................................Box 222, Tecumseh, Nebr.
CLAPP, L. R...Silverton, Colo.
CLARK, FREDERICK P......................4311 Barring Ave., East Chicago, Ind.
CLARK, HOPEWELL..........................1027 Summit Ave., St. Paul, Minn.
CLARKE, ROY H...Takilina, Josephine Co., Ore.
COOPER, M. D......................Care Ford Collieries Co., Curtisville, Pa.
CORWIN, FRANK R., Asst. Supt., Care Consolidated Arizona Smelt. Co.,
Humboldt, Ariz.
COUNSELMAN, THEODORE B.............808 Sellwood Bldg., Duluth, Minn.
COXE, EDWARD H......Care T. W. Guthrie, 1735 Oliver Bldg., Pittsburgh, Pa.
CRISPELL, C. W..............................55 W. Chestnut St., Kingston, N. Y.
CROSTON, JOHN J...Box 46, Victor, Colo.

Cunningham, George H., Constr. Engr., Electrolytic Plant,
Anaconda Copper Mining Co., Great Falls, Mont.
Dalburg, F. A.........................Casilla 324, Lima, Peru, South America.
Daniel, William B., Care Frank W. Daniel, Comly St. & Delaware River,
Wissinoming, Philadelphia, Pa.
Davies, Fred A.............................304 Pine St., Anaconda, Mont.
Dawson, Thomas W., Min. Engr., Chief Engr., H. C. Frick Coke Co., Scottdale, Pa.
Deane, Willard Albert, Flotation Engr., Research Dept., Anaconda Copper Co.,
Anaconda, Mont.
Dickman, R. N.............................1010 Maelers Bldg., Chicago, Ill.
Durham, E. B.............Care Mammoth Copper Mining Co., Kennett, Cal.
Eaton, E. R.........Care Kirby Thomas, Esq., 120 Broadway, New York, N. Y.
Edelen, A. W..Socorro, N. Mex.
Edmondson, H. W..............................Silver Mine, Madison Co., Mo.
Eldredge, R. B..Ward, Colo.
Ellis, Hubert I...Jordan Valley, Ore.
Fraser, William L., Supt., Oro Grande Min. Co.,
737 Citizens National Bank Bldg., Los Angeles, Cal.
Friedrich, Alfred K.........................3226 W. 14th St., Cleveland, O.
Gay, Ware B.....................Mayo Land & Bridge Co., Richmond, Va.
Gibbons, F. J....................Great Western Copper Co., Courtland, Ariz.
Graff, W. W...Box 155, Gwinn, Mich.
Graham, Walter F......................................Painted Post, N. Y.
Greenan, J. O......................805 Crocker Bldg., San Francisco, Cal.
Gregory, Edward Henry, Min. Engr......P. O. Box 157, Guadalajara, Jal., Mex.
Hale, H. C...........The Hale-Stephen Co., 500 Schofield Bldg., Cleveland, O.
Hamlyn, J. J.........................Yuba Construction Co., Marysville, Cal.
Hansen, C. A..................Care General Electric Co., Salt Lake City, Utah.
Head, James L..............................703 S. Clark St., Moberly, Mo.
Heizer, O. F.........................70 Black Oak Mine, Saulsbyville, Cal.
Herman, Hyman, Geological Survey of the Mines Department,
Melbourne, Vict., Australia.
Hess, K. F., Flotation Foreman......Burro Mountain Copper Co., Tyrone, N. Mex.
Hill, Frank Albert.........................411 Mutual Life Bldg., Seattle, Wash.
Hinckley, E. R..................Care The Royal John Mine, Sherman, N. Mex.
Hines, P. R., Care Mining Machinery Dept., Allis Chalmers & Co., Milwaukee, Wis.
Hocking, R. O..................................Box 667, Nashwauk, Minn.
Hoffmann, John S..Hill City, S. Dak.
Holmes, R. G., Salesman, The Denver Rock Drill Mfg. Co.,
508 First Avenue South, Seattle, Wash.
Hook, J. S..Ponca City, Okla.
Hursh, Robert.......Care The Empire Zinc Co., 703 Symes Bldg., Denver, Colo.
Hutchins, J. P..................Marskaya 21, Apartment 24, Petrograd, Russia.
Hyde, James M..Silverton, Colo.
Imhoff, Wallace G..........................6805 Penn Ave., Pittsburgh, Pa.
Ingalsbe, F. R...............................Forest Service, Missoula, Mont.
Ishihara, K..........Kenjibo Iron Works, Mitsubishi Co., Okaido, Chosen, Korea.
Jahn, William F...Riebeling, Mont.
Jensen, Joseph..Gallup, N. M.
Johnson, F. A., Care Standard Oil Co. of N. Y., Changsha, Hunan Prov., China.
Kane, John I., Care American Smelt. & Ref. Co., 1112 Mills Bldg., El Paso, Tex.
Keep, Glenn A.............................1624 E. 73d St., Cleveland, O.
Kellogg, L. O..........................629 W. 135th St., New York, N. Y.
Kemp, L. W.....................................Instructed to hold everything.
Kennedy, George A..........................527 Symes Bldg., Denver, Colo.
Klopstock, Paul......Paul Klopstock & Co., 17 Battery Place, New York, N. Y.
Koch, Hugo E., Bench Chemist, Atlas Portland Cement Co.,
Care Y. M. C. A., Hannibal, Mo.
Kohl, E. William, Jr...........Central Borjita, Dos Caminos, Oriente, Cuba.
Kong, S. T., Engr., Hupeh Government Mining Bureau, Wuchang, Hupeh, China.
Krogh, Alvin R.............................304 Pine St., Anaconda, Mont.
Lamb, R. B..............................25 Broad St., New York, N. Y.
Lamb, Richard, Cons. and Constructing Engr.......90 West St., New York, N. Y.
Leland, Everard...Clifton, Ariz.
Levis, A. C...Box 678, Golden, Colo.
Lewis, Robert S.................The Pines, Bass Lake, P. O., Madera Co., Cal.

Libbey, F. W. .... 307 North First St., Phoenix, Ariz.
Lipscomb, Robert A. .... Covington, Va.
List, E. .... Care Nevada Smelting Co., Nevada, Mo.
Loo, P. C. .... 214 W. Newton St., Boston, Mass.
McBride, W. G. .... 2027 East First St., Long Beach, Cal.
McCrodan, B. A., Chemist, Old Dominion Copper Mining & Smelting Co., Globe, Ariz.
McFadden, George Caldwell .... 407 West Main St., Danville, Ill.
McKechnie, B. E. .... Hanover Bessemer Iron & Copper Co., Fierro, N. M.
Merriman, T. C. .... 249 Ellsworth Ave., New Haven, Conn.
Miller, C., R., ... Engr., Graham Bros., Stockholm, Sweden.
Moran, Robert B. .... 208 Union League Bldg., Los Angeles, Cal.
Morrison, George A., Min. Engr., Care A. E. Morrison, 1983 W. 111th St., Chicago, Ill.
Nebeker, William G. .... Alta Club, Salt Lake City, Utah.
Nibecker, Karl .... Mulberry and Spruce Ave., N. W., Cleveland, O.
O'Brien, Thomas S., Care Joshua Hendy Iron Works, 75 Fremont St., San Francisco, Cal.
Oliver, Earl .... 115 E. 15th St., Bartlesville, Okla.
O'Neil, Frederick W. .... 432 Newberry Blvd., Milwaukee, Wis.
Orrok, George A. .... R. F. D. 3, Willimantic, Conn.
Overpeck, A. C. .... Kingman, Ariz.
Pearl, Holman I .... Bruce, Wis.
Pettis, E. S. .... Chichagof, Alaska.
Pittman, Frank L. .... Contact, Nev.
Place, Richard G. .... 1105 Garfield Ave., Pasadena, Cal.
Plumb, E. T. .... 11 Polhemus Place, Brooklyn, N. Y.
Pollen, Clement H., Care N. A. Wallinger, Esq., Cranbrook, B. C., Canada.
Pope, John D. .... 829 West Quartz St., Butte, Mont.
Powell, F. .... 5011 15th Ave., N. E., Seattle, Wash.
Power, F. Danvers, Perpetual Trustee Chambers, 33-39 Hunter St., Sydney, N. S. W., Aust.
Pratt, Morton E. .... Casilla 28, Guayaquil, Ecuador, South America.
Prescott, Basil .... 3314 Douglas St., El Paso, Tex.
Raiber, Nicolaus H. .... 147 South Hyde Park Ave., Scranton, Pa.
Rainsford, R. S., Care J. C. White & Co., Inc., 43 Exchange Place, New York, N. Y.
Ralph, Walter H. .... Care U. S. Mines, Bingham Canyon, Utah.
Raymond, R. W. .... Washington, Litchfield Co., Conn.
Rhoads, Albert E. .... Y. M. C. A., Denver, Colo.
Riddell, Guy C., Cons. Met., Broken Hills Associated Smelters Proprietary Co., Ltd., Port Pirie, N. S. W., Aust.
Rigg, Gilbert, Care Associated Smelters Proprietary, Ltd., Collins House, Collins St., Melbourne, Aust.
Roberts, Edward J., Daley Cons. Mines Co., 27 W. 3d St., Salt Lake City, Utah.
Ross, H. W. .... Care Backus & Johnson, Lima, Peru, South America.
Rossbach, E. J., Sales Engr., Sullivan Machinery Co., Suite 413, 122 South Michigan Ave., Chicago, Ill.
Schiffner, Otto E. .... Great Western Cons. Min. Co., Box 503, Tonopah, Nev.
Seibert, P. A. .... Clearspring, Md.
Semple, R. A., Care The Federal Dyestuff and Chemical Co., Kingsport, Tenn.
Shaffer, W. B .... Lock Box 62, Nazareth, Pa.
Shaver, Herbert H. .... Shavertown, Luzerne Co., Pa.
Sherman, Fred W. .... Ray Cons. Copper Co., Ray, Ariz.
Simon, Trevor B. .... 5736 Hobart St., Pittsburgh, Pa.
Skerritt, A. W., Care H. W. Davidson, Esq., Chosen Gold Mines, Ltd., Seoul, Chosen, Korea.
Smith, E. Percy .... Hailey, Idaho.
Smith, Frank A. .... Care P. Paulson, Rawlins, Wyo.
Smith, Lyon .... Gladstone Hotel, Chicago, Ill.
Snow, Frederick W., Care Robert Williams & Co., Friars House, New Broad St., London, E. C., England.
Soper, Ralph H. .... 307 Central National Bank Bldg., Tulsa, Okla.
Starkey, Tom R., Lieut., C Co. 1st Lakour Battalion, R. E., British Expeditionary Force, Care General Post Office, London, England.
Starr, Charles C., Mgr., Gage Tungsten Property & Field Engr., Goldfield Cons. Mines Co., Deming, N. M.

STAVER, W. H. .... 1808 Rivermont Ave., Lynchburg, Va.
STRUTHERS, JOSEPH, Chem., Met.; Treas., Federal Export Corpn., 111 Broadway, New York, N. Y.
THOMPSON, A. P .... Care Green Monster Min. Co., Jerome, Ariz.
THOMAS, D. R .... Moose Mountain, Ltd., Sellwood, Ont., Canada.
THORNHILL, E. BRYANT .... Metals Recovery Co., Cobalt, Ont., Canada.
UDE, EDGAR .... 3601 So. Grand Ave., St. Louis, Mo.
VAN ZWALUWENBURG, A., Care Walter Harvey Weed, 29 Broadway, New York, N. Y.
VOORHESS, FREDERICK A .... 2814 Ellendale Place, Los Angeles, Cal.
WALTER, E. W .... Silverton, Colo.
WALTERS, M. B .... Instructed to hold everything.
WATTON, EUGENE E., Pres., The United Oil Co., 309 McPhee Bldg., Denver, Colo.
WEIR, GEORGE, North Broken Hill, Ltd., Collins House, Collins St., Melbourne, Vict., Australia.
WENSLEY, ROGER L., Care William Young Westervelt, 17 Madison Ave., New York, N. Y.
WHITE, EDWIN ELLIS, Supt. of Mining & Prospecting, Tata Iron & Steel Co., Ltd., Sakchi, India.
WHITE, HAROLD E .... 516 Elmwood Ave., Buffalo, N. Y.
WHITE, J. L., Supt. of De Soto Mine, Cons. Arizona Smelt. Co., Ocotillo, Ariz.
WILLARD, JAMES O .... 211 S. Dithridge St., Pittsburgh, Pa.
WILMOT, H. C .... Santiago, Chile, South America.
WILSON, FRANK LAKE .... 437 63d St., Oakland, Cal.
WILSON, M. O .... 120 Broadway, New York, N. Y.
WITT, HERBERT N .... Goldfield, Nev.
WONG, S. C .... 526 N. School St., Honolulu, H. I.
WRIGHT, PERCY E .... 5533 33d Ave., N. E., Seattle, Wash.
YOUNG, HAYES W .... Asst. Prof. of Met., Stanford Univ., Cal.

MEMBERS' ADDRESSES WANTED

Name.	Last address of Record from which Mail has been Returned.
BARNES, BLAKESLEE	Arrow Engineering Co., Palmyra, Mo.
BELL, D. A. S	136 McLaren St., Ottawa, Canada.
BLOW, J. J	172 Rodney St., Brooklyn, N. Y.
BOOTH, E. L	349 W. 145th St., New York, N. Y.
BOYS, H. R	Aurora Cons. Min. Co., Aurora, Nev.
BRENNAN, J. C	Tequisquiapan, Queretaro, Mexico.
CAIRNS, John M	Henry S. King & Co., 65 Cornhill, London, E. C., England.
COLE, ROBERT J	McKay Apartments, 7th & Pike Sts., Seattle, Wash.
COOK, PAUL RICHARDSON	4159 Grand Boulevard, Chicago, Ill.
CRARY, CHARLES N	Kimberly, Nev.
DRAPER, CARL H	Apartado 77, Guadalajara, Jal., Mexico.
FOSTER, GEORGE C	18 Cadogan Blk., Calgary, Alta., Canada.
GOEDICKE, CARL	Box 535, San Antonio, Tex.
GORDON-FIREBRACE, W. E	812 Salisbury House, London, E. C., England.
GRANT, ULYSSES S., IV	Westmorly Court, Cambridge, Mass.
GUNTHER, C. GODFREY	Stratford, Conn.
HANLON, JOHN EDWARD	Timmins, Ont., Canada.
HOBART, EDMUND NORRIS	Clifton, Ariz.
JONES, THOMAS J., Mine Mgr., Kyshtim Min. Wks., Perm Govt.,	via Petrograd, Russia.
KISHMAN, MAURICE W	101 Masonic Ave., Cripple Creek, Colo.
LENOIR, FRANK H	Douglas, Alaska.
MCGEE, JOHN	United Greenwater Co., Dale via Amboy, Cal.
MAINWARING, H. M.C., Mine Mgr.,	Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MILLER, FRANK BARTON	Blair, Esmeralda Co., Nev.
MOHRMAN, E. M	1293 W. 111th St., Cleveland, O.
MOORE, REDICK R	Zortman, Mont.
O'BRIEN, P. C. K	Hotel Essex, 684 Larkin St., San Francisco, Cal.
PARRISH, S. F	Battle Mountain, Nev.
PATCHELL, F. J	4516 No. Lincoln St., Chicago, Ill.
PATERSON, A. W	1814 11th Ave., Spokane, Wash.
PERRY, ROBERT S	Kingscourt Apts., 36th & Chestnut Sts., Phila., Pa

RALPH, E. W.......Boston Ely Min. Co., Kimberly, Nev.
REVELL, G. E.......Box 132, Nelson, B. C., Canada.
REYNOLDS, L.......Address wanted.
RHODES, W. B.......Golden, Colo.
RODRIQUEZ, J. C.......Apartado 87, Saltillo, Coah., Mexico.
ROGERS, B. C., Care Standard Mine, Detroit Copper Min. Co., Box 427, Metcalf, Ariz.
SALES, A. J.......Giroux Cons. Mines Co., Kimberly, Nev.
SANDIFER, H. C.......Box 15, Bis., Mexico City, Mexico.
SMITH, A. H.......Gateway, B. C., Canada.
SPARKS, J. T.......326 Erie St., El Paso, Tex.
STODDART, A. W.......Address wanted.
SULLIVAN, W. P.......The Henry Walke Co., Norfolk, Va.
TAYLOR, A. W.......Korean Exploration Co., Chiksan Mines, Chiksan, Korea.
THOMAS, EDMUND.......Weaver Mine, Gibson, N. Mex.
TONG, S. K.......413 W. 115th St., New York, N. Y.
VAN NESS, W. W.......Address wanted.
WAINEWRIGHT, W. B.......61 Grace Church St., London, E. C., England.
WENTWORTH, I. H.......245 Beldon Ave., Harlandale Addition, San Antonio, Tex.
WRAIGHT, E. C.......63 Wavertree Road, Streatham Hill, London, S. W., England.
YEANDLE, W. H., JR.......Address wanted.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period May 10, 1916 to June 10, 1916.

Date of Election.	Name.	Date of Decease.
1899	**Burt, Andrew	1916.
1915	*Chandler, J. W	May 7, 1916.
1875	DuBois, A. J.	1916.
1883	Hill, Frank A	July 13, 1915.
1910	*Parrish, Robert L	July 23, 1915.

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* Member. ** Life member.

Andrew Burt, of Elphinstone Collieries, Scotland, was killed in active service on the European battlefield.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.
DAVID H. BROWNE, *Chairman.* PERCY E. BARBOUR, *Vice-Chairman*
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer.*

Boston

Meets first Monday of each winter month.
W. E. C. EUSTIS, *Chairman.* R. L. AGASSIZ, *Vice-Chairman.*
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.
STANLY A. EASTON, *Chairman.* FREDERIC KEFFER, *Vice-Chairman.*
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
D. C. LIVINGSTON, FRANK A. ROSS.

Puget Sound

Meets second Saturday of each month.
GLENVILLE A. COLLINS, *Chairman.* H. L. MANLEY, *Vice-Chairman.*
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
L. F. LAUCKS. JOHN N. POTT.

Southern California

SEELEY W. MUDD, *Chairman.* C. COLCOCK JONES, *Vice-Chairman.*
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Bldg., Los Angeles, Cal.
RALPH ARNOLD. A. B. W. HODGES.
A. B. CARPENTER. WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman.* F. H. BOSTWICK, *Vice-Chairman.*
P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
G. A. KENNEDY. M. S. MacCARTHY

Montana

J. L. BRUCE, *Chairman.* W. C. SIDERFIN, *Vice-Chairman.*
M. H. GIDEL, *Secretary-Treasurer*, 1102 W. Galena St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.
T. A RICKARD, *Chairman.* W. H. SHOCKLEY, *Vice-Chairman.*
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
E. A. HERSAM. H. W. YOUNG.

Pennsylvania Anthracite

R. V. NORRIS, *Chairman.*
CHARLES F. HUBER, *Vice-Chairman.* EDWIN LUDLOW, *Vice-Chairman.*
W. J. RICHARDS, *Vice-Chairman.* ARTHUR H. STORRS, *Vice-Chairman.*
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI. *Chairman.* HERMAN GARLICH, *Vice-Chairman.*
F. W. DeWOLF, *Vice-Chairman,* M. M. VALERIUS, *Vice-Chairman,*
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH,

Chicago

CHARLES H. MacDOWELL, *Chairman.* LUTHER V. RICE, *Vice-Chairman.*
HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

Utah

C. W. WHITLEY, *Chairman.* WALTER FITCH, *Vice-Chairman.*
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI. WILLIAM WRAITH.

Arizona

GERALD SHERMAN, *Chairman.*
NORMAN CARMICHAEL, *1st Vice-Chair.* B. BRITTON GOTTSBERGER, *2nd Vice-Chair.*
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
W. L. CLARK, J. C. GREENWAY.
W. G. McBRIDE. FOREST RUTHERFORD.

STANDING COMMITTEES

Executive

L. D. RICKETTS, *Chairman.*

GEORGE D. BARRON,
SIDNEY J. JENNINGS,
W. L. SAUNDERS,
BENJAMIN B. THAYER.

Membership

KARL EILERS, *Chairman.*

ARTHUR S. DWIGHT,
LEWIS W. FRANCIS,
LOUIS D. HUNTOON,
ARTHUR L. WALKER.

Finance

GEORGE D. BARRON, *Chairman.*

ALBERT R. LEDOUX,
CHARLES F. RAND.

Library

E. GYBBON SPILSBURY, *Chairman.*[3]

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[1] Until Feb., 1917. [2] Until Feb., 1918. [3] Until Feb., 1919.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Stoping in the Calumet and Arizona Mines, Bisbee, Ariz.

BY PHILIP D. WILSON,* B. S., E. M., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

The mines of the Calumet & Arizona Mining Co. are situated in the Warren Mining District, Cochise County, Arizona, between Bisbee and Warren and adjoin those of the Copper Queen Consolidated Mining Co. and the Shattuck Arizona Copper Co. Paleozoic limestones of considerable thickness have been intruded by granite porphyry. A great stock of the intrusive outcrops boldly near the city of Bisbee and is known as Sacramento Hill. From this central core the porphyry has tongued out into the limestone for many thousands of feet as dikes and sills of irregular shape and variable dimensions. Churn drilling has recently developed a large tonnage of secondarily enriched copper ore in the main porphyry mass of Sacramento Hill. While some ore has been found in porphyry in other parts of the district it is relatively unimportant compared with that which occurs in the Paleozoic limestone as irregular lenses replacing favorable beds near porphyry contacts or associated with well-marked fracture zones often many hundreds of feet away from the intrusive.

In a zone extending from Sacramento Hill for a distance of from 1,500 to 3,000 ft. the limestone has been extensively metamorphosed and in this contact metamorphic zone the ore occurs through a known vertical thickness of upward of 900 ft. and individual orebodies often show extension in a vertical direction of over 200 ft. At a greater distance from the intrusive stock the ore occurs as lenses of greater horizontal than vertical dimensions with an average thickness of approximately 30 ft. An exception to this rule is found in the occasional steeply dipping mineralized fractures, almost true tabular veins, which are found long distances away from the zone of contact metamorphism.

The copper was probably introduced during or slightly later than the porphyry intrusion in the form of chalcopyrite and bornite. These primary copper minerals are always associated with pyrite and often occur as rich lenses in a huge mass or shell of pyrite carrying an insignificant percentage of copper. In many instances the copper-bearing solutions were preceded by siliceous solutions and these latter, often persisting during the period of mineralization, have thoroughly indurated the orebody and the surrounding country rock. Oxidation took place during two distinct periods separated by a period of submergence in the Cretaceous sea and subsequent elevation, distortion and tilting of the sediments. This has resulted in a most irregular zone of oxidation. In some cases enriched chalcocite ore and even thoroughly oxidized ore are found stratigraphically far below primary orebodies. Where oxidation has been complete the carbonates and oxides of copper occur beneath or entirely surrounded by a mass of oxides of iron and aluminum, often of great extent. While there has been considerable enrichment with development of chalcocite in the district, opinion, in view of recent developments of the primary ores, is tending toward the conclusion that migration of values has been for the most part local and that both the oxidized and the chalcocite ores depend for their present copper content rather on rich primary sulphides than on any extensive leaching and subsequent enrichment of low-grade material. The large masses of iron and aluminum oxides found in connection with the oxidized ores are then the oxidation products of the shells of lean pyrite and pyritic limestone which surround the primary copper sulphides rather than gossans from which appreciable copper has been leached.

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Conditions Determining Stoping Methods

The great variety of ore occurrences in the district and the equally variable character of the country rock in the vicinity of the orebodies render any uniformity of stoping method impossible where the factors of economy and efficiency are considered. In the early days of Bisbee all of the ore mined was taken from the irregular high-grade oxidized bodies found within a comparatively short distance of the surface. Square-set stoping with subsequent filling was used with much success and even today this time-tried method of Philip Deidesheimer is recognized as the system most widely applicable to the Bisbee ores. It was not until comparatively recent years that the natural reluctance to try new methods while the old could still be used was overcome, in the Calumet and Arizona mines under the superintendence of W. B. Gohring, with resultant reduction of costs in some cases to less than one-half of those formerly obtained.

The considerations determining the stoping method to be used may be summarized as follows:

1. Safety and efficiency of working conditions for the men.
    - (*a*) Physical characteristics of ore and country rock.
    - (*b*) Ventilation.
2. Maximum ultimate profit.
    - (*a*) Cost of mining.

  - (*b*) Ore which may be economically sacrificed.
  - (*c*) Relative economy of mining and of sorting out included waste.
  - (*d*) Rapidity with which it is advisable to mine, or daily tonnage required.
3. Shape and dimensions of orebody.
4. Safety of mine.
  - (*a*) Cost of timber.
  - (*b*) Availability of filling.
  - (*c*) Fire risk.

SQUARE-SET STOPING

The square-set system is easily the most flexible and where the orebody is very irregular and has large included blocks of waste it is the most satisfactory. It is a simple matter to leave the waste behind as a portion of the filling, and where the mining of too large a section at once is not attempted and filling is kept within a reasonable distance of the back it is as safe or safer than any of the other methods in vogue. By this method the cost* of stoping per ton including labor, powder, timber, carbide and air, ranges from $0.80 in sulphide ore to $1 or even $1.30 in oxide ore. Where the ground is very heavy, additional timber as bulkheads and double-up sets increases the cost materially and in sulphide ore the forest of timber makes the risk of fire an important consideration. In a normal square-set stope the item of timber, which laid down at the mine costs about $17.50 per thousand, amounts to 25 per cent. of the total.

Recovering Square-set Stope Timbers

The first attempt to reduce this expense was made some years ago by M. W. Mitchell, foreman of one division of the Calumet and Arizona mines, who devised a method for "robbing" a considerable portion of the timber from a square-set stope during the process of filling. Approximately 50 per cent. of the timber can be recovered where conditions will permit of this robbing process, and the actual stoping cost based on a series of stopes over a considerable period in which the timber has been recovered and used again is reduced by about 8 per cent. While applicable to a large majority of square-set stopes, in many instances the ground is too heavy to risk removing the timbers and in some cases the timber is so badly damaged that it is not worth extracting. The method finds its greatest economy where an orebody is being mined in successive sections so that the recovered timber may be left in open square sets at the edge of the stope and used when the adjacent section is being worked, as the cost of excessive handling soon reduces the saving to a minimum.

When an attempt to rob the timbers from a square set is anticipated the first section is carried up five sets wide, as shown in Fig. 1. The length of the section is determined by the character of the ground and varies from five to as many as ten sets. It is advantageous to carry the section as long as the ground will permit, since the longer the section the greater will be the ultimate timber recovery. It is usually not feasible to attempt to recover timber from a stope over 50 ft. in height. When an orebody, the thickness of which is greater than this, is to be mined and the timber recovered by this method the operation is carried on in successive lifts, as will be hereinafter described. In stoping the section a gangway is maintained in the central row of sets on the sill floor and chutes, *ee*, built in each alternate set on either side. Slides at appropriate places in the stope will deliver the broken ore to these chutes as stoping proceeds, with a minimum of shoveling. The square sets are so erected that the caps, *bb*, lie parallel to the gangway and the girts, *cc*, consequently at right angles to it. When the ore has all been extracted and the stope is ready for robbing, 6 by 6-in. braces, *dd*, are placed between the two diagonally opposite caps in the second and fourth rows of sets on the two floors above the sill, as shown in Section *AA*, Fig. 1. If the stope is taking weight badly a brace may be necessary at each end of each of these caps, but in some cases a single brace in the middle of the cap is sufficient. It is usually the case where robbing is attempted that the plans include the

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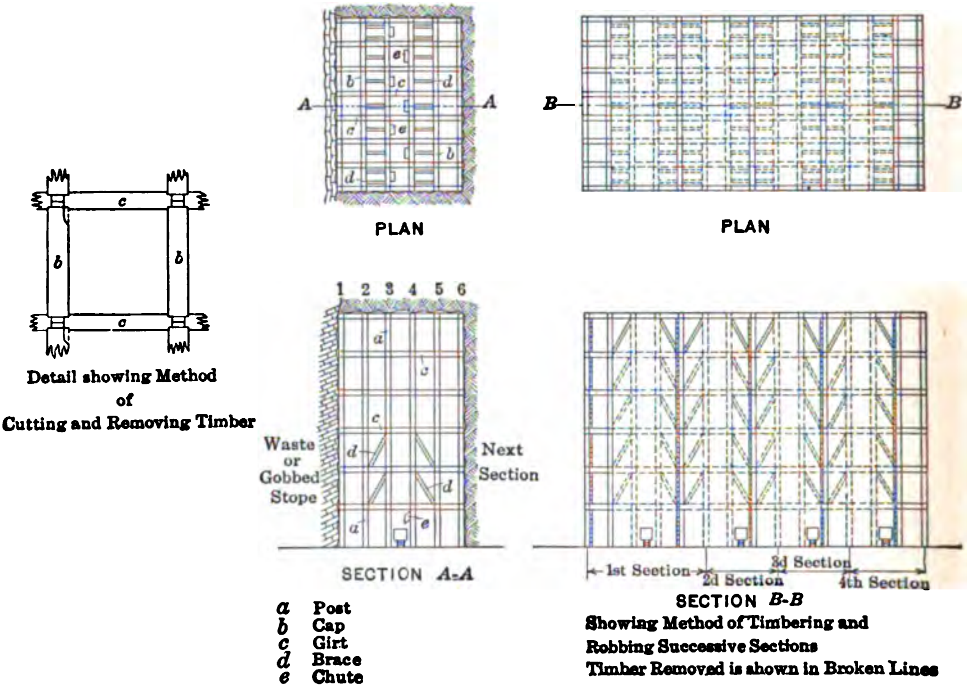
* These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers. During the past year the sliding wage scale and the abnormally high cost of supplies have resulted in a material increase in all mining costs.



Detail showing Method of Cutting and Removing Timber

Showing Method of Timbering and Robbing Successive Sections
Timber Removed is shown in Broken Lines

FIG. 1.—MITCHELL METHOD OF RECOVERING SQUARE-SET STOPE TIMBERS.

robbing of successive sections which will each be mined in turn after the preceding one has been robbed and filled. If such is the case, gob lagging is spiked to the rows of posts 1 and 5, Section *AA*, so that when filling is introduced the gob will lie tightly against the ground or previously gobbed stope on that side of the section, in this case the left, where no further mining is anticipated. One run of sets on the other side of the section will be left open when the stope is filled, so that the timber as extracted may be stored in these sets and used when the adjacent section is carried up.

After the braces have been put in place the caps in row 2 are cut, as shown in the detail, Fig. 1, to permit of slipping out the girts between rows 2 and 3. When these have been removed it is a simple matter to remove the caps and the sill and first floor posts in rows 3 and 4 and the girts between rows 3 and 5. Waste filling is then introduced until it reaches to within a few inches of the second floor above the sill, the caps cut and the timber on this floor between rows 2 and 5 removed as before. The braces, partially buried in the gob, may be drawn out by throwing one end of an ordinary mule skinner's tail-chain over the exposed end of a brace and fastening the other to the end of a piece of timber. Using this piece of timber as a lever and the cap still in place above as a fulcrum the brace is easily loosened and drawn from the filling. During each downward movement of the lever the chain automatically loosens and slips lower on the brace, so that one man can easily manage the operation. The braces as removed are placed diagonally between the caps at the top and bottom of the third floor above the sill. These braces may obviously be used again and again until weakness or damage impairs their usefulness. The process as described with alternate filling and drawing timbers a floor at a time is repeated until the top of the section is reached and the stope filled. The timber as removed is stored in the open run of square sets for use in the adjacent section. In all cases the posts and girts on each end of the stope are left behind and if the ground is unusually heavy it is safer to leave an entire run of sets to support the ends of the stope as the timber in the center is being removed. These are left in the gob so that the ultimate timber recovery is much reduced. The braces, as the timbers below are removed, take up the vertical and part of the lateral pressure, but the major portion of the latter is taken up by the gob. It is essential, therefore, that the filling be kept well up to the level of the floor from which the timber is being removed.

If the ground is unusually heavy or the orebody over 50 ft. in thickness the section must be carried up in successive horizontal lifts in order that there may be a minimum of ground open at any one time before filling is introduced. It is the usual practice to open the section three sets high, brace, remove the timbers and fill as before the lower two floors. With the stope thus securely reinforced, the section may be carried two sets higher when the two lowest open floors are robbed and filled as before. If the ground is so heavy that additional precautions are necessary the timber is first drawn from only one end of the stope and filling is introduced into the robbed portion while the adjacent row of timber on the same floor is being extracted. Thus, as soon as the timber is removed, it is replaced by gob so that the walls of the stope are at all times amply supported. Using this method of mining and recovering the timbers in successive lifts, there is virtually no limit to the height to which a section may be carried safely in heavy ground.

The adjacent section and each subsequent one are carried only three sets wide, but the same length as before. The method of timbering and robbing successive sections is shown in Section *BB*, Fig. 1. In each case the stope is carried up alongside of the run of square sets left open in the previously mined section so that four sets of ground are actually open. The timber extracted from the preceding stope is used as far as it goes in the new section. That which is in the best condition is used where it can be recovered again and the partly damaged material where it will be left behind in the gob. When the robbing process begins in a new section the braces are placed in the run of sets left open in the previously mined section and the center run of the new one. The caps cut are those directly against the gob. In every case it is the caps to be left behind that are cut, leaving unmutilated all of the timber to be removed. As filling is introduced, gob lagging is spiked as shown to the outside of the center run of sets, leaving one run open as before for storing timber for the next section. It is evident from Section *BB*, Fig. 1, that in using this method in several successive stope sections only every third row of posts and caps is left behind in the gob while all the intermediate timbers are recovered. The secrets of success in using this method are the narrow stopes and care in keeping the filling well up to the floor from which the timber is being taken.

MITCHELL SLICING SYSTEM

Mr. Mitchell is also responsible for the next improvement in stoping methods which has found wide application in the Calumet and Arizona mines during the past few years. This method, known as the Mitchell slicing system, has been described in an article by M. J. Elsing in the *Engineering and Mining Journal*, July 23, 1910, but as several changes and improvements have been made in it since that time it will bear further mention. It was arrived at more or less by accident. A large block of ore broke away from the back while a heavy sulphide stope was being worked and in order to recover it, long stringers were thrown across the top of the ore to support the back and the ore mined from above by underhand stoping. The method as finally developed from this simple beginning is applicable to orebodies in which the hanging wall is fairly flat and regular and the lateral pressure not too great. It is flexible, however, by virtue of the fact that square sets may be used in conjunction with the method in mining irregular and outlying portions of the orebodies. Furthermore, there must be no large quantity of waste in the ore, for while it is possible to sort and leave some waste material behind in the stope the method loses many of its advantages if careful sorting must be done.

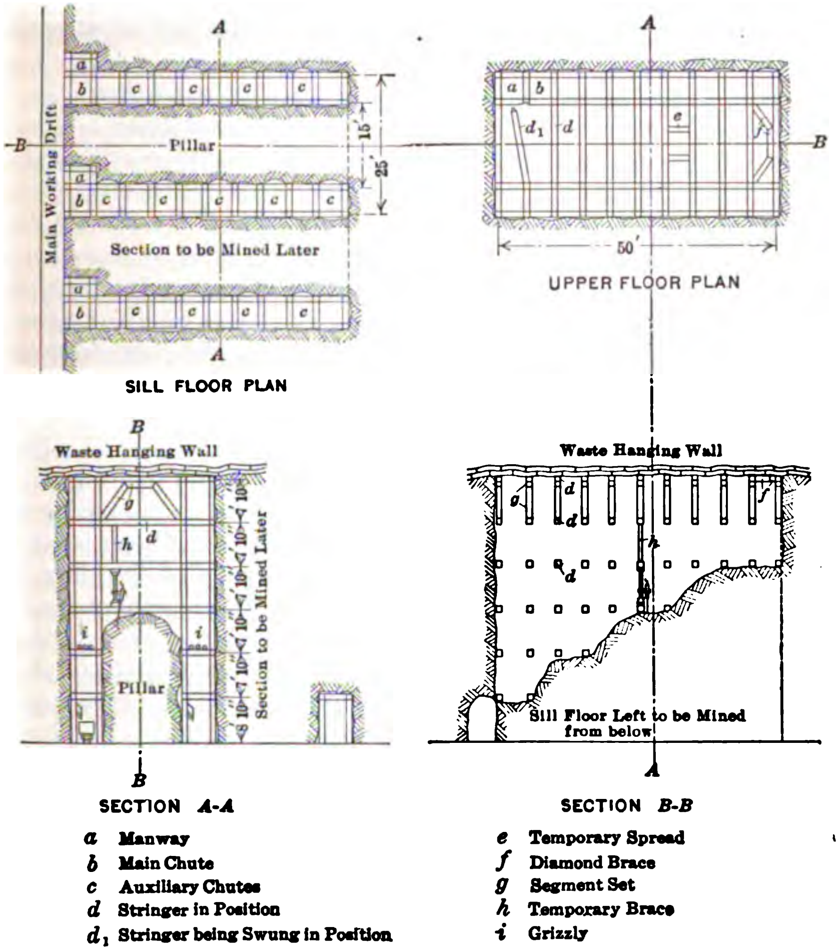


FIG. 2.—MITCEELL SLICING METHOD.

After preliminary prospecting has roughly outlined the shape and dimensions of the oreshoot, two lead rows of sill square sets are driven, 20 ft. center to center, at right angles to the general strike of the body from the main working drift to the end of the section to be mined, as shown in the sill floor plan, Fig. 2. The length of the section varies from 25 to 50 ft., depending upon the character of the ground. Where it is possible to run cars directly into these two crosscuts a small sloping cut is made in every alternate set on the same side of each crosscut for a loading chute. Care must be taken to keep these cuts small and under no circumstances to take them on both sides of the same pillar, for if the base of the pillar is weakened trouble is sure to ensue. If it is not practicable to load the cars in these side crosscuts, as is often the case, the first set alone may serve as a chute and the cars be loaded directly in the main drift. A six-post raise is carried over the first two sets in each cross-cut through to the level above to be used as a timber and manway and later as a waste chute. On the sill floor a small manway, *a*, is cut out next to the chute set as shown, and on the floor above the ladderway is carried up over the chute while the other compartment of the raise is lined for a chute. When the raises have holed to the next level runs of square sets are carried directly over the lead rows of sill sets to the top of the ore, leaving a pillar of ore 15 ft. wide and the length of the section resting on its own base to be subsequently mined in horizontal slices from the top downward. Under normal conditions standard 10 by 10-in. square-set timbers are used in these preliminary operations, but if the ground is unusually heavy 12 by 12-in. timbers are advisable.

The first slice is taken, as shown in Sections *AA* and *BB*, Fig. 2, over the whole top of the pillar, cutting it loose from the waste hanging wall. Machines are first erected in the topmost square sets and holes pointed in such a manner that the ore passes with a minimum of shoveling onto grizzlies in the open run of square sets. Where only one chute is used the broken ore fills the sets until its normal angle of repose is reached, the remainder running from the chute by gravity. The 10 by 10-in. stringers, *dd*, 15 ft. long and framed with 2-in. tenons, are thrown across the open slice between the caps of the topmost run of square sets and directly under the hanging wall with such blocking and lagging as is necessary to keep the stope safe and the timbers secure. In placing a stringer in position the tenon on one end is inserted in the place between caps and post framed to receive it in the same manner as an ordinary girt. The opposite cap is cut 2 in. to allow the tenon on the other end of the stringer to be swung into position, as shown in the upper floor plan, Fig. 2, and a piece of lagging spiked to the cap to hold it in place. The topmost stringers are supported temporarily by stulls resting on ore in the pillar until enough ground has been broken to permit of laying similar stringers between the caps of the next lower run of square sets. When these stringers, resting on the unbroken ore in the pillar, have been placed in position the stulls are replaced by segment sets, *gg*, of 10 by 10-in. timber, erected, as shown in Section *AA*, Fig. 2, to support the upper stringers and the back of the stope. While 10 by 10-in. or 8 by 10-in. stringers are necessary directly under the hanging wall, it is the usual practice to use 8 by 8-in. timber for those subsequently put in where the weight to be carried is much less. Segment sets are usually unnecessary except below the topmost stringers, although occasionally used below if the ground is very heavy.

The next slice is then taken across the top of the pillar. Either jackhamers or piston machines set up on columns between the stringers are used to drill down holes, pointed so that the ore is broken directly into the chutes. In subsequent slices, if a piston machine is used, it is usually necessary to put in temporary 6 by 6-in. braces, *hh*, between the stringers vertically above the drill column in order to make the machine secure while drilling. It is sometimes safer to put in temporary horizontal spreads, *ee*, between the stringers near the holes which are about to be blasted. If the ground at the end of the stope begins to swell or work 10 by 10-in. diamond braces, *ff*, are placed in position as shown in the upper floor plan, Fig. 2, to take up longitudinal pressure. The slices are taken to within one floor of the sill leaving one set of ore to be mined from below. A floor is then laid over the entire stope and 2-in. gob lagging is spiked vertically to the inside of both vertical runs of square sets, leaving the square sets open when the stope has been filled to be used as one set of chutes in each of the adjoining sections. If the ground is very heavy it has been found impracticable to leave these sets open, and in that case they are lagged on the outside and filled with the rest of the stope. The filling is introduced through the two raises from the level above and if the stope is not taking weight badly a considerable proportion of the stringers and segment sets, if the latter have been used, may be recovered as filling progresses. Upward of 50 per cent. of the stringers may be thus removed under favorable conditions. If it has been considered advisable to leave the square sets open the timber recovered may be stored in these sets and used when the adjoining section is taken. Where the timber cannot be used in a stope in the immediate vicinity it is usually better economy not to attempt to extract it, as much handling soon neutralizes any saving gained.

The maximum vertical height of a section which may be safely mined by this method has been found to be about 60 ft. When the ore extends from level to level, or 100 ft. thick, the stope is worked in two lifts of 50 ft. each, the upper lift first. In this case raises are driven through from level to level as before but the preliminary rows of square sets are driven 50 ft. above the sill and carried from there to the upper level. An attempt was made at first to carry these runs of sets the entire 100 ft., but it was found that by the time the upper lift was mined out and filled the timbers below were so badly crushed as to be useless. The upper lift is mined as has been described with the exception that the slices are taken for the entire 50 ft. before the floor is laid and filling commences. It is obvious, too, that all of the ore must perforce be drawn from only two chutes instead of from the series of chutes as is optional in the method as originally devised.

Where waste material must be sorted from the ore and left behind, flooring may be laid on the stringers and a certain amount of waste piled on the floors. A recent adaptation of the method, which eliminates to a certain extent this makeshift device, depends on mining the pillars in benches as shown in Section *BB*, Fig. 2. The slices are taken out more rapidly in the end of the stope nearest the waste raises so that the lowest slice is mined out, the floor laid and filling brought in while mining is still progressing on the upper benches at the other end of the stope. When this stage is reached any waste encountered may be mixed with the filling in the worked-out end of the section. Mucking broken ore is further reduced by this improvement as the ore runs more readily into the chutes if the benches are properly taken.

In preparing to work a section adjoining a filled stope, a new lead row of sill sets is driven, as shown in the sill floor plan, Fig. 2, leaving a 15-ft. pillar as before, the vertical run of square sets carried to the hanging wall, and this run of sets used in conjunction with the one of the other stope which is still open, if it has been possible to so leave it, as chutes in working the new section. Thus but one run of square sets with the consequent narrow work is necessary for each section. If it has not been possible to leave the square sets open in the filled section it becomes necessary to take the slices at such an angle that the broken ore will run by gravity into the new run of sets. The bench method is particularly useful in this regard as it facilitates the gravity handling of the ore.

The saving in labor, timber, and powder by this method over the square-set stope is an obvious and substantial one, but the main advantage of the system is the rapidity with which the ore may be mined if necessary and the increased tonnage yield per man in the stope. While 5 or 6 tons per man is considered creditable in a square-set stope, it is not considered extraordinary in mining the pillars in the Mitchell system to reach three or four times those figures and in auger ground a yield as high as 50 tons per man per shift has been attained. The actual saving in stoping cost as shown in the appended table amounts to from 20 to 30 per cent. of that of square-set stoping.

MITCHELL TOP-SLICE CAVING SYSTEM

It was decided about two years ago that the top-slice caving method as used on the Minnesota iron range, in Cananea, and elsewhere, might be applied to a large body of oxide ore in which heavy and swelling ground made the cost of square-set stoping excessively high. The fact that there was no working level above this ore from which the stopes might be filled also contributed to the choice of the top-slice system. Although the method as usually practised worked with some degree of success and reduced the cost of stoping substantially, Mr. Mitchell, who was in charge of the work, was not satisfied with the results, and from it he evolved an inclined top-slice caving method which reduces the large amount of handling of ore in the stope, incidental to ordinary top-slicing, to a minimum.

The method is applicable to the same conditions as those under which ordinary top-slicing is successful, that is to say, uniform ground which is too heavy for economical square-set stoping and which will cave readily as required, but may be at all times kept under adequate control. It may often be used with increased economy in ground adapted to square-set stoping if the other conditions obtain. Before deciding upon this method there should be reasonable assurance that there is no ore left unextracted above the body to be caved, as the subsequent mining of this

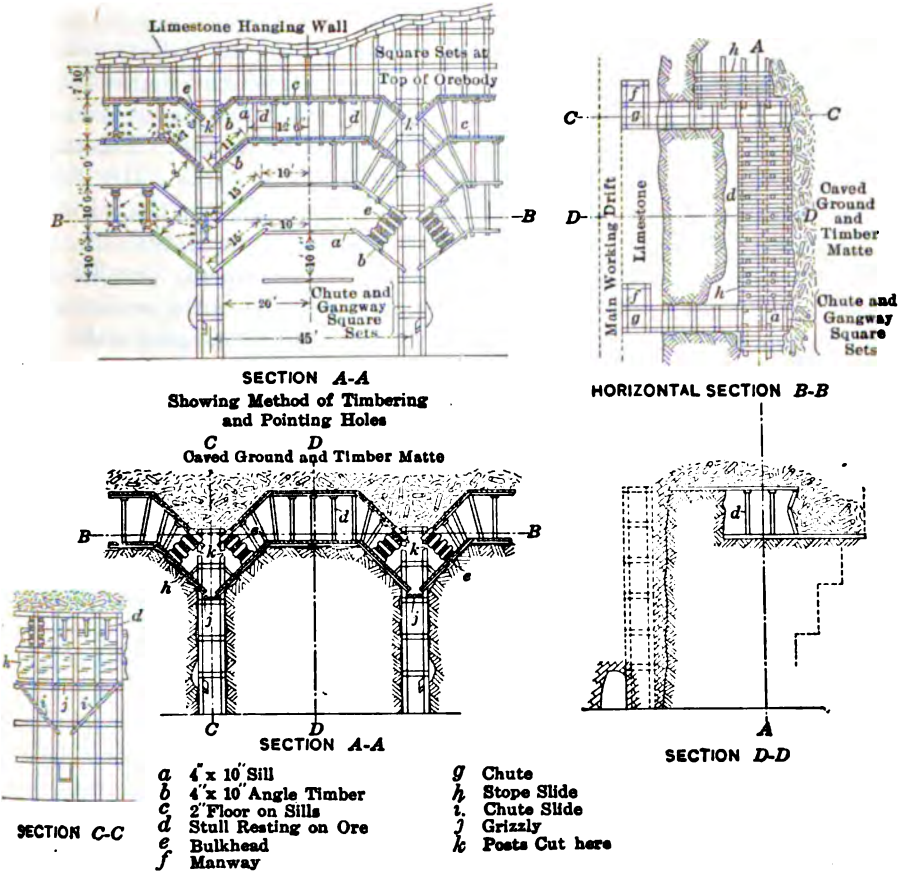


Fig. 3. Mitchell Top-slicing Caving System.

ore would be attended by many difficulties after the country has been well broken up. In case development work is being done on a level above that of the stopes, this method, unlike square-setting, affords no means of disposing of the development waste, which consequently must be trammed to the shaft and hoisted. In common with ordinary top-slicing, ventilation is poor in the stopes worked by this system. There is sure to be some ore sacrificed in the process even under the most satisfactory conditions, which is an important consideration in the Warren District where a comparatively high-grade ore is mined.

In preparing for this method, the orebody is laid out in 45-ft. sections at right angles to its general strike. These sections may be mined the width of the ore to a maximum of about 80 ft. and 100 ft. is the economic limit of the height to which each section should be carried. Permanent six-post raises are driven, as shown in Sections *AA*, *BB* and *DD*, Fig. 3, preferably in waste at the edge of the ore, to the elevation of the top of the orebody. One compartment of each is used as a chute, *g*, the other as a timber and manway, *f*. From the chute compartment, parallel lead rows of square sets are carried the width of the ore, or of the section to be taken, on the three floors directly beneath the waste hanging wall. On each of the floors below these the square sets are carried a distance decreasing as the sill is approached leaving a foot wall of unbroken ground sloping in the direction of the main drift on which the broken ore will run directly into the chutes. This leaves a pillar 40 ft. wide which is mined by horizontal slices from the top downward, one-half from either side simultaneously, and the back allowed to cave as stoping progresses.

Before taking the first slice, the ground on either side of the top row of square sets is broken by holes drilled from a set-up in the square sets in such a way that 4 by 10-in. angle timbers, *bb*, 7 ft. 6 in. long, may be rested on solid ground at an angle of about 45°, as shown in Section *AA*, Fig. 3; 2-in. flooring is laid on these angles, making a slide, *h*, upon which the subsequently broken ore will run by gravity upon grizzlies, *jj*, in the square sets and thence into the chutes. The first cut is taken from both sides simultaneously across the pillar at the end of the ore or section farthest from the main drift and is timbered with standard square sets. The slice then proceeds from this cut toward the main drift, working from both sides of the section at once and supporting the waste back with square sets; 4 by 10-in. sills, *aa*, cut to lengths and with bevels as shown in Section *AA*, Fig. 3, are laid at the base of the posts, furnishing a support for the upper end of the angle timbers. Beneath the beveled joints of these sills pieces of 2-in. plank are spiked to render them rigid and to act as headboards for the stulls, *dd*, placed under them when the next slice is taken; 2-in. flooring is laid on top of these sills. As the slice retreats toward the main drift the rows of square-set posts first erected are shot out and the back caved, leaving at all times at least two open sets between the working face and the caved ground. To prevent the vertical run of square sets from being crushed by the weight of the caved ground, small bulkheads, *ee*, are built under the lower caps of this floor resting upon the angle timbers. The posts are then cut at points, *kk*, directly under the angles, thus transferring the weight from the square sets below to the solid ground in the pillar.

When the upper slice has proceeded far enough from the extremity of the section so that the ground first caved is solid, preparations may be begun for the second slice. A machine is set up in the open run of square sets and ground broken so that 4 by 10-in. angle timbers 11 ft. long may be laid as before at a slope of about 45°. The angle timbers and sills above are caught up as shown with round stulls resting on the unbroken ore in the pillar, 6 to 8 in. in diameter as the ground may require. The preliminary cut is carried as before about 5 ft. wide and 9 ft. high across one-half the width of the section, or 20 ft., erecting stulls under the sills beneath the caved ground and timber matte above; 4 by 10-in. sills are so laid that they do not come vertically below the upper sills, for the vertical stulls supporting the upper sills should rest directly on solid ground. These sills are covered with 2-in. flooring as before. The erection of drill columns and the pointing of holes is shown in Section *AA*, Fig. 3. The breast holes shown are usually unnecessary, as the cut holes and lifters are in almost all cases sufficient to break the ground. After the preliminary cut has been taken the major portion of the ore is slabbed directly into the chute sets, leaving from one-third to one-fifth to be mucked. Stulls should be just strong enough so that the third stull back from the working face is commencing to crush as the face is being drilled, as shown in Section *DD*, Fig. 3.

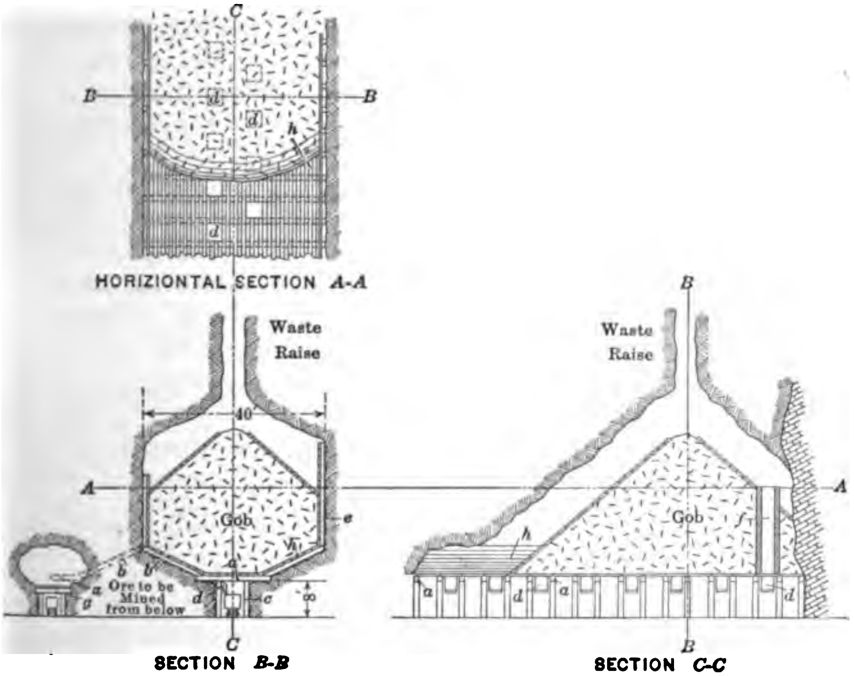
As slicing proceeds downward the lower floors of the vertical runs of sets must be extended to maintain the slope of the solid ground to the extreme end of the section and permit the broken ore to run readily into the chutes. Care must be taken not to take out the vertical runs of sets too far ahead of operations, for the weight on them is considerable even if the greatest precautions are taken. If the orebody is over 80 or 100 ft. in width and consequently too wide to be mined in one section at right angles to its strike, it is not advisable to attempt to leave a vertical wall of ore against the caved ground and timber matte at the end of the section. In this case each new slice is started about 5 ft. closer to the working raises than the slice above, as shown in broken lines in Section *DD*, Fig. 3. Thus the end of the stope is left in steps and the danger of losing ore which may break from the wall and mix with the matte is eliminated. The ore so left may be recovered readily when the next section is taken. As mining approaches the sill floor, chutes may be put in the sill sets, as shown in Section *AA*, Fig. 3, at such intervals as will facilitate the rapid and efficient drawing of the broken ore. Bulkheads, with subsequent cutting of posts to take the weight from the vertical run of sets, need not be erected oftener than conditions require. In normal operations bulkheads in every third slice afford sufficient protection to the square sets below. Enough partly crushed timber comes through from the timber matte above to build these bulkheads so that their cost is insignificant. It has been found that the maximum thickness of slice which can be safely carried under the conditions encountered in the Calumet and Arizona mines is 10 ft. 6 in. and slices of this thickness are not attempted until the fourth slice is taken and the caving matte is under perfect control. If waste is encountered in the ore it is quite feasible to sort it out and leave it behind to become incorporated with the caved ground and timber matte, but like the method last described the system loses many of its advantages if careful sorting must be done.

As shown in the appended table, a 10 per cent. reduction in stoping cost is gained by this method over ordinary top-slicing. The labor cost is cut about 15 per cent. and the item of powder is substantially reduced. The ore may be mined rapidly and safely in the heaviest kind of ground. An output of about 10 tons per man per shift or a total of about 125 tons per day can be maintained from each section.

Gilman Cut-and-Fill System

When the large bodies of primary sulphides of great vertical as well as horizontal extent, from which about one-half of the total tonnage of the Calumet and Arizona mines is now being derived, were first opened up they were mined by ordinary shrinkage stopes of great size, the dimensions being determined solely by the limits of the orebody. When the stope was finished the shrinkage ore was drawn out and waste filling introduced. While confined to virgin ground this method was quite safe and satisfactory as the country rock was strong massive limestone and the walls sound. As more of the country was opened, however, these large open holes became dangerous and the method evolved itself into a combination of shrinkage and cut and fill with virtually no limit to the width of the stope or the amount of open ground above the gob. But increasing disturbance of the country rock as mining proceeded recently made it necessary to develop some safer method of mining these great sulphide lenses without too great a sacrifice of the very low costs and flexibility of the systems previously used. Consequently a cut-and-fill method was devised by Oscar Gilman, foreman of one division of the Calumet and Arizona mines, which has remedied the existing evils of the older systems and still permits of a minimum of timber, low costs, great flexibility and large daily tonnage. The method may be used wherever the ore and country rock are strong and solid and, while the width of each stope section taken is limited to 40 ft., the length and height of stopes are limited only by the extent of the commercial ore, although for convenience sections are usually included between levels 100 ft. apart vertically.

Crosscuts are driven at 40-ft. intervals through the orebody from a strike drift, preferably in country rock in one wall of the orebody, at right angles to its general strike. A cutting-out stope is started, as shown in Section *BB*, Fig. 4, by enlarging the crosscut with stoping drills. When it has reached a height of about 8 or 9 ft. above the rail, hitches are cut in the walls and temporary so-called "Cousin Jack" stringers, *gg*, about 10 ft. long are thrown across the crosscut at 5-ft. intervals 5 ft. above the rail; 4 by 10-in. timbers are used at each side of the crosscut to support these stringers where the ground requires them. Flooring is laid upon the stringers in a direction parallel to the crosscut so that ore subsequently broken may be loaded directly into cars run into the crosscut through the opening made by removing one of the center planks. This device is used only during the early stages of the operation to eliminate mucking from the floor of the crosscut before the stope has reached proportions large enough to permit of the erection of permanent sill timbering and chutes. The cutting-out stope is then enlarged by drilling deep holes



SECTION *B-B*

SECTION *C-C*

a Sill Stringer
b Rafter Stringer
c Post
d Chute
e Stull
f Cribbed Chute
g "Cousin Jack" Stringer
h Slide

FIG. 4.—GILMAN CUT-AND-FILL SYSTEM.

with stoping drills, maintaining an arched back and a slope of about 20° on the two sides of the crosscut to a width of 40 ft. Hitches are cut in the sides of the stope to receive 16-ft. by 12-in. by 12-in. stringers, *aa*, which are laid at 5-ft. intervals for the length of the section 8 ft. in the clear above the rail. The "Cousin Jack" stringers are then removed and if the ground requires it the 16-ft. stringers are supported by 10 by 10-in. posts, *cc*, 5 ft. in the clear. This leaves ample room for the 42-cu. ft. motor cars into which the ore is loaded in these crosscuts. As a rule, no braces are necessary between stringers but where required 4 by 10-in. timbers are used. Then 18-ft. by 10-in. by 10-in. rafter stringers, *bb*, beveled as shown in Section *BB*, Fig. 4, are laid on the sloping sides of the stope with one end resting on each 16-ft. stringer; 4-in. flooring is laid over the whole and chutes, *dd*, built on both sides of the drift in each alternate set. A 10 by 10-in. brace is used between the stringers at the side of the crosscut where the chute is to be built.

While this preliminary preparation has been going on, at about the center of each section a raise has been started which is ultimately connected with the level above to be used for timber and waste filling. If waste is encountered in driving this raise it is discontinued until filling is needed in the stope. To provide for absolute safety no more than 10 ft. of open ground is left at any one time between the floor and the back of the stope. The initial enlargement of the stope takes place, as shown in Sections *BB* and *CC*, Fig. 4, beneath the waste raise, and when the 10-ft. limit has been reached, waste is admitted and forms a cone extending out toward the edges of the stope. Temporary flooring is laid upon this gob at its normal angle of repose leaving only 2 or 3 ft. between the floor and the back of the stope. The back may be supported in any weak spots while the next slice is being taken by temporary stulls or bulkheads resting upon the waste. As the chutes under this gob pile are covered over they are abandoned and the ore as broken runs upon the sloping floor and is drawn from the chutes still open at either end of the waste.

As mining exposes the sides of the stope, 8 by 10-in. stulls, *ee*, from 8 to 16 ft. in length are placed vertically at 5-ft. intervals, resting on the rafter stringers. Where one stull is placed upon another as the stope progresses upward they are securely lashed together, on four sides if possible, and each tied into the gob with timber "dummies." It is advisable to use as long stulls as practicable for every joint is a point of weakness when the next section is taken; 2-in. gob lagging is spiked horizontally to these vertical stulls before filling is admitted. In the actual process of mining it is often found that a few long holes judiciously placed toward the sides of the stope will when blasted break the arch and break a much greater tonnage than would normally be expected. When the limits of the orebody approach the vertical it becomes necessary to carry up a chute or two, *ff*, next the waste wall with 4 by 10-in. cribbing through the waste filling, to be used as ore chutes. The length of the stope often becomes so great as mining progresses upward that one raise will not serve for filling the whole stope. In this case another raise may be driven through to the level above, or the outlying portions of the stope filled by driving small inclined raises off into the waste walls. It is apparent, however, that under normal conditions waste raises may be a long distance apart.

Comparative Stoping Costs[1]

Method and Conditions	Labor	Explosives	Timber	Candles or Carbide	Air	Other Supplies	Total
Square-set, oxide ore,[2] heavy ground	$0.73	$0.06	$0.34	$0.01	$0.12	$0.04	$1.30
Square-set, oxide ore, average ground	0.60	0.05	0.25	0.01	0.10	0.04	1.05
Square-set, oxide ore, robbing timbers	0.61	0.05	0.16	0.01	0.10	0.04	0.97
Top-slice caving, old method, oxide ore, heavy ground	0.63	0.07	0.20	0.01	0.10	0.03	1.04
Mitchell top-slice caving, oxide ore, heavy ground	0.54	0.04	0.21	0.01	0.10	0.03	0.93
Mitchell slicing, oxide ore, average ground	0.51	0.05	0.20	0.01	0.06	0.02	0.85
Square-set, sulphide ore,[3] average ground	0.49	0.04	0.19	0.01	0.04	0.03	0.80
Mitchell slicing, sulphide ore, average ground	0.43	0.03	0.15	0.01	0.03	0.01	0.66
Cut-and-fill, Gilman method, sulphide ore, good ground	0.34	0.04	0.07	0.01	0.04	0.01	0.51
Cut-and-fill, old method, sulphide ore, good ground	0.32	0.04	0.04	0.01	0.04	0.01	0.46

[1] Figured on the basis of the wet tons mined. These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers.

[2] Oxide ore in place will average from 12 to 16 cu. ft. to the ton.

[3] Sulphide ore will average from 9 to 10 cu. ft. to the ton.

If there is a great demand for ore, stoping may be commenced in the adjoining section while still mining in the original section. If this is contemplated it is better practice to drive the original waste raise nearer one end of the section so that there may be a considerable thickness of gob at that end while stoping is still in progress in the lower portion of the other end. Stoping in the new section is carried on in identically the same manner as before. The new waste raise is driven near the same end of the section as in the adjoining partially filled stope so that mining in the new section proceeds most rapidly adjacent to that part of the original stope in which the gob is thickest and most firmly consolidated. In mining alongside of this old filling care must be taken not to expose too much of it at one time, and the stull joints must be watched. But if proper precautions are taken no difficulties are experienced in mining two or more parallel stopes simultaneously. It is inadvisable to leave a section with the idea of mining it after the stopes on either side have been finished and filled. Carrying a stope by this method between two gobbed sections is at best an uncertain operation and the risk may be entirely eliminated by taking the sections in succession as described. The ridge of ore 35 ft. wide and 14 ft. high between each stope on the sill floor is left to be mined from below. While as yet no stopes have been carried up to this point, no difficulty is anticipated as the two rafter stringers braced against each other will be virtually self-supporting and in any event it will prove a simple matter to catch them with stulls from below.

Phenomenally low costs for the Warren District have been obtained with this system of stoping. It will average about $0.50 per ton and a much lower figure has been reached under especially favorable conditions. Although slightly higher than the cost of the old shrinkage method the added consideration of safety more than counterbalances the small increase over the more hazardous system. The stope ventilation is good and the efficiency of the men high. An average yield of about 12 tons per man per shift can be maintained under normal working conditions. The daily tonnage output of each section will average 100 tons and may be crowded to 150 or even 200 tons per day.

SUMMARY

In comparing the various methods described, the considerations enumerated in the early part of the paper must be applied to each system. The cut-and-fill system is easily the cheapest and most satisfactory in every way where conditions will permit of its use. The item of timber, usually a large one, is much reduced and the risk of fire is virtually eliminated. While any large horse of waste encountered in the ore may be kept separate and left in the gob, any attempt at close sorting is inadvisable. As in the case of the Mitchell slicing system, if the ore is fairly clean the greatest ultimate economy will be gained by mining it all as it comes without attempting to sort. The consequent low stoping cost obtained by mining large tonnages rapidly will more than counterbalance the increased cost of handling and treating a slightly lower grade material. In neither system can the grade of ore be controlled as closely as in the square-set or top-slice caving methods. Where the ore is very irregular or badly mixed with waste the square-set method is still found to be the most satisfactory. If the stope is in good condition a considerable proportion of the timbers may be safely extracted as it is filled.

For a heavy uniform orebody the Mitchell inclined top-slice caving system is a good one. While it still retains some of the disadvantages of the old-fashioned top slice it is in many ways a great improvement over the older method. Its principal advantages are the substantial saving in labor and powder, and the greater rapidity with which the ore may be mined. Its unfavorable features are those of all top-slice caving methods, the almost inevitable sacrifice of some ore, narrow work, poor ventilation in the stopes and the fact that the ground above is so badly broken as to render the expense of future work on upper levels so great as to be in many cases prohibitive. The Mitchell slicing system finds its chief application to fairly regular bodies in not too heavy ground. Under favorable conditions lower costs are obtained with this method than by any of the others in use, with the exception of the cut-and-fill method. Its saving in labor and timber is a very appreciable one over square-set stoping and the cycle of preparing, mining, filling and abandoning a section is normally a very short one. This feature, in addition to permitting a large daily output, reduces considerably the repair cost incidental to keeping a section of the country open over a long period. In short, the introduction of these methods has effected a tremendous saving to the Calumet & Arizona Mining Co. during the past few years without the slightest sacrifice of the safety of the men or of the mines in which they have been used.

In closing the author wishes to acknowledge his indebtedness to M. W. Mitchell and Oscar Gilman, mine division foremen of this company, for invaluable assistance in the accumulation of data for this paper.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Flotation of Minerals

BY ROBERT J. ANDERSON,* B. S., ROLLA, MO.

(Arizona Meeting, September, 1916)

INTRODUCTION

During the past 5 years no subject has aroused more interest or received more attention among mill operators than flotation. One reason for this is, undoubtedly, the remarkable success of the process in Australia where that country has risen to the position of furnishing one-fifth of the world's supply of zinc. The plant of the Butte & Superior Copper Co., designed by James M. Hyde in 1912, was the first important large flotation plant in this country; other installations have been made since then with considerable frequency, particularly so in the last 2 years.

Without going into complete detail, it may be safely said that although flotation is a highly successful commercial process, it is more or less on an empirical basis. Upon scanning the literature, it is found that the flotation investigations have, in the main, dealt with a solution of the problems which accrue to practise; until recently no attempt was made to remove the difficulties in the way of the formulation of a consistent and harmonious theory. Of late this phase of the work has been receiving considerable attention, particularly by the United States Bureau of Mines at its Salt Lake City station, by the Mellon Institute at Pittsburgh, by the General Engineering Co. at Salt Lake City, by many of the mining schools, and private individuals almost *ad infinitum*. There is, at present, an enormous amount of laboratory work being carried on in flotation research, and probably the time is not far away when a generally acceptable explanation of flotation phenomena can be set forth. However, at this time, it seems as though the current technology is befogged with a superabundance of contradictory evidence; this confusion can only be dispelled as time proceeds and the knowledge of the subject becomes more complete.

Many phenomena are supposed to contribute to the flotation of minerals, whether in whole or in part is a mooted question. I shall only sketch roughly the present tendency of ideas and make no reference to the first early and crude notions which have little value other than historical interest. Many questions have arisen in connection with flotation concerning such matters as surface and interfacial tensions, adsorption, absorption or occlusion, colloids, emulsions, electrolytic and electrostatic phenomena, etc. These phenomena are to be discussed in later paragraphs in order to ascertain what bearing they may have, if any, on froth flotation. No final solution of this very large problem is attempted in this paper, for in any discussion of flotation theory, at this time, one is confronted by so many obstacles that at best only a rather fragmentary presentation can be made. An attempt has been made in this writing to correlate the operative and fundamental principles and compare in a measure some of the theories so far advanced.

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Discussion of Certain Factors

Surface and Interfacial Tension

Surface tension has been rather well defined in articles appearing in the *Journal of the American Chemical Society* during the years from 1908 to 1913. The theory of surface tension has been treated in particular by Laplace, Gaus, and more recently by Van der Waals, and by Willows and Hatschek.[1] As defined by Jones,[2] "potential energy, present at the surface of liquids, produces a tension which is known as surface tension." The phenomena which are invariably indicative of surface tension are commonly observed: Drops of a liquid not exposed to an external force, *i.e.*, either suspended in another liquid of the same specific gravity or freely falling, assume a spherical shape, the sphere being that form of body with the smallest surface per given volume; further, if water be placed in an open vessel its surface film will be a measurable quantity, and its thickness will vary with a number of factors of which temperature is one. Its thickness is observed as ranging from 4×10^{-5} cm. to 4×10^{-8} cm., and its density, when referred to the main bulk of the water below, will approximate 2.14. Surface tension is not affected by the surface area. It is numerical in value and expressed in dynes per centimeter. It is a variable factor dependent on temperature, increasing numerically with falling temperature, *e.g.*, water at 18° C. has a surface tension of 73 dynes per centimeter, and at 0° C. 75 dynes per centimeter. At the critical temperature of a liquid its surface tension becomes nil.

All liquids have a definite cohesion or tensile strength, which is ascribed to the well-known mutual attraction of their molecules. This then is comparable to a pressure existing within a liquid, which has been termed the "intrinsic" pressure. Naturally the value of the surface tension of solids is numerically a high one. The surface tension of a pure liquid against its vapor can be and is markedly affected by the addition of soluble contaminants. Some salts will raise the surface tension of water while others will lower it; the fact that the salts of weak acids will lower the surface tension of water is explained by the fact that free acid is liberated by hydrolysis. It is further known that all acids will lower the surface tension of water. The surface tension of water is decreased by the addition of oil, or better, oil will reduce the interfacial tension between the water-air phases. A phenomenon for which no explanation has been given is the one which shows that the addition of contaminants may either raise or lower the surface tension of water, but such addition, while it may decrease that tension greatly, can increase it only slightly. Any lowering of surface tension is more marked in a liquid which has a high surface tension, *e.g.*, water, than in liquids of low surface tension.

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[1] Willows and Hatschek: *Surface Tension and Surface Energy*, 1915.

[2] Jones: *Elements of Physical Chemistry*, 1907.

There can be, of course, no surface tension without adsorption, which produces, in the case of positive adsorption, an increased surface concentration resulting from a lowering of the surface tension by the contaminating and dissolved substance whatever it may be. The equation of Gibbs—$u = -\, c/Rt \cdot d\sigma/dc$—gives the relationship between surface tension and the distribution of the solute between the bulk of the liquid and the film interface. Here the notation is:

u = excess of substance in the surface layer,
c = concentration in the main body of the liquid,
R = the gas constant,
t = absolute temperature,
σ = surface tension.

This shows that when the surface tension is reduced by the addition of a contaminant, the quantity $d\sigma/dc$ is negative and u is positive (from algebraic consideration). The surface film then contains more of the contaminant than the main body of the solution. If the surface film contains less of the contaminant than the main body of the solution it is a case of negative adsorption.

As given in the foregoing, the surface of a liquid against its vapor is in tension—surface tension; the surface of liquid against another liquid, or a gas or solid, is also in a state of tension: this is termed interfacial tension. In the consideration of the bearing which surface and interfacial tensions have on froth flotation, this condition of affairs obtains in the flotation machine: Pulp consisting of ore of approximately 80-mesh, water in ratio of 3:1 of ore, and oil in disappearingly small amount, is being violently agitated. For the sake of a specific case, the air is being forced mechanically into the swirling pulp by beaters or stirrers. These phases then are present in flotation by the oil-froth process, viz., solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air), liquid-liquid (water-oil), liquid-gas (water-air), and liquid-gas (oil-air). Thus six tensions are present, but if the oil is soluble in the water the tension number becomes three. It is known that pure water can not be made to maintain a persistent froth because its surface tension is too high. Acid, if present, will lower the surface tension of water, as will oil if it is soluble to any extent. Very many interesting and important, as well as speculative, consequences follow from a consideration of interfacial and surface tensions, and; in a later paragraph, we shall have occasion to return to the consideration of these complicated phenomena.

Certain metallic sulphides, *e.g.*, galena, have the power of floating on undisturbed water; they are not wetted and the curve of contact is convex. Some gangue minerals, *e.g.*, quartz, possess an adhesive force of attraction for water which exceeds the intrinsic pressure of the water; they are therefore wetted and sink to the bottom, being drawn through the surface film. Such properties of the minerals are affected by the presence of oil, acid, and other reagents. Oil has a greater adhesive attraction for sulphide minerals than for gangue minerals; and the addition of acid and oil (if it is soluble) acts as a contaminant which will lower the surface tension of the water and aid in the production of a persistent froth. Let us now look into the question of adsorption and see what part it plays in flotation, since it is so requisite to the production of a variable surface tension.

Adsorption

Generally speaking, adsorption deals with the unequal distribution of substances at the interface between dissimilar phases such as, solid-solid, solid-liquid, solid-gas, liquid-liquid, liquid-gas, and gas-gas. It is purely a physical effect. Commonly, adsorption[3] is construed to be the result of the condensation of a disperse phase upon the interfacial boundary solid-liquid. Returning for a moment to the Gibbs equation mentioned above, adsorption may occur if the interfacial tension solid-liquid is reduced—this being positive adsorption. If, however, such an interfacial tension is raised in value it is a case of negative adsorption as the solute or disperse phase will be rejected from the surface. Any condensation, strictly stated, of a solute or disperse phase in the interfacial boundary separating liquid-liquid or liquid-vapor is held to be a special case of adsorption. However, in the general sense, the phenomenon is looked upon as being the result of condensation of a disperse phase in the interface of two immiscible phases. Adsorption is shown very strikingly by colloid gels—the product obtained by the coagulation of sols—and certain cases of selective adsorption are very remarkable. Adsorption will naturally vary with the surface exposed. In Miss·Benson's experiments with amyl alcohol in aqueous solution, amyl alcohol reduced the surface tension of the water, and it was found by producing a voluminous froth that the alcoholic concentration in the froth exceeded that in the bulk of the aqueous solution by about 5 per cent. A froth has a very large surface, and it would be expected that the adsorption would be greater. Such experiments prove the value, qualitatively, of the Gibbs rule.

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[3] Briggs: *Journal of Physical Chemistry*, vol. 19, No. 3, p. 210 (March, 1915).

Recent work shows that all solids do condense certain amounts of gases on their surfaces and retain them there with very great tenacity. Liquids in like manner adsorb gases. Further, liquids and solids exhibit marked selective adsorption of gases. Although this selective adsorption obtains, no evidential proof has been submitted which says that the amount of gas adsorbed by one substance is largely different than the amount adsorbed by another substance. An electric charge on an adsorbed substance probably would considerably influence the amount adsorbed. The adsorption of air plays an important rôle in flotation, for as Breuer points out, the adsorbed air film is enormously responsible in preventing the coalescence of solid particles.

A comprehensive study of the adhesion of small particles of solid to the dineric interface (surface separating two liquid phases) has been made by Hofmann[4] based on the theory of Des Coudres.[5] From the standpoint of flotation this may be given as follows: If a solid particle, *e.g.*, quartz, is wetted much more strongly by water than by another liquid, *e.g.*, oil, the water will displace the oil, and a film of water will form about the quartz particle according to the relative forces of adhesion. Then the quartz particles will remain in the water phase if the water has a specific gravity greater than the oil, regardless of their size; but if now the oil has a greater specific gravity than the water, then the quartz particles will remain in the water phase until the size of the particles is such that the force of gravity will remove them from the water. Conversely, if a solid particle, *e.g.*, galena, is wetted more strongly by oil than by water, the oil will form a surface film about the particle and hence prohibit the particle from being wetted by water, *i.e.*, from entering the water phase. Then the galena will only enter the water phase when the water is more dense than the oil, and, further, when the galena particles are of such a size that the force of gravity overcomes the adhesion of the oil film to the oil.

Returning to purely theoretical considerations, Hofmann draws certain conclusions at this juncture which deal with the supposition that solid particles will then remain in the surface separating two immiscible liquids, if those particles are wet partially by each liquid. I quote Bancroft at length on this matter:[6] "The solid particles tend to go into the water phase if they adsorb water to the practical exclusion of the other liquid; they tend to go into the other liquid phase if they tend to adsorb the other liquid to the practical exclusion of the water; while the particles tend to go into the dineric interface in case the adsorption of the two liquids is sufficiently intense to increase the miscibility of the two liquids very considerably at the surface between solid and liquid."

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[4] *Zeit. Phys. Chem.*, vol. 83, p. 385, 1913.

[5] *Arch. Entwicklungsmechanik*, vol. 7, p. 325, 1898.

[6] Bancroft: *Journal of Physical Chemistry*, vol. 19, No. 4, p. 287 (April, 1915).

Any simultaneous adsorption of two immiscible liquids by a solid would probably tend to result in the formation of a homogeneous liquid phase at the surface of the solid.

In regard to the effect of contaminants or other impurities in contact with two immiscible liquids, this condition obtains: If the contaminant is soluble in one liquid but not in the other, and also lowers the interfacial tension of the two, the equation set forth by Gibbs exacts the requirement that the contaminant should obtain in the interface. Examples of this prove the validity of the law.

The terms adsorption and absorption have been used interchangeably in some writings, thus contributing to the already existing confusion of ideas.

Absorption or Occlusion

There are three ways by which gases can be held with reference to solids: viz., (1) By surface adsorption; (2) in solid solution; and, (3) by occlusion. The term "occlusion" has been applied indiscriminately to any of the above methods by which gases are held by solids. Strictly speaking, by occluded gas is meant gas which is absorbed and held in finely divided pores or openings which may be of microscopic size. A recent theory[7] holds that occlusion plays the operative rôle in the flotation of minerals by all processes. I am unable to reconcile myself to this explanation for a number of reasons. Marked instances of occlusion at normal temperature are known only in certain amorphous substances like charcoal. Many metals, of course, both in the liquid and solid states, have the power of occluding gases, often in marked degree. There may be and undoubtedly are fine pores in the floatable minerals, which may in a sense be considered as an assemblage of capillary tubes; these can and do occlude gas. Yet occlusion is marked only in amorphous substances and in certain metals as just stated. It is definitely known that occluded gases are retained with very great tenacity by the substances occluding them and therefore expelled only with difficulty. It seems anomalous then to hold that the occluded gas can depart from the mineral occluding it with sufficient speed to aid the air bubbles in the liquid in the process of flotation. I believe rather firmly that occlusion is not a cogent factor in flotation by any process, and that a more consistent theory may be formulated without postulating these conjectures regarding occlusion.

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[7] Durell: *Mining and Scientific Press*, vol. 111, No. 12, p. 428 (Sept. 18, 1915) and Durell: *Metallurgical and Chemical Engineering*, vol. 14., No. 5, p. 251 (March 1, 1916).

Colloids

Colloids, in the original definition of the term by Thomas Graham, are not a definite class of substances; rather a large number of different substances may be made to assume the colloidal state if proper precautions are taken. All of which reveals the striking fact that this colloidal condition is a *state* and not a *form* of matter. The ultra-microscope of R. Zsigmondy and H. Siedentopf has increased the knowledge of colloids to a great extent. A rather general statement may be made regarding colloids and that is, that they do not show osmotic pressure in any appreciable amount. Colloidal solutions—sols—are regarded as systems of two phases, in which the dissolved substance is the disperse phase and the solvent the continuous phase.

Since in flotation the ore is often as small in size as certain of the colloids, the flotation pulp (ore, water, etc.) can be looked upon as a coarse suspension, and the laws of colloids apply here with equal force as in the realm of colloidal chemistry. So-called suspensions are systems consisting of solid particles of microscopic size distributed through a liquid. As mentioned by Ralston,[8] Reinders has treated at length the particular case of a solid phase maintained in contact with two liquid phases, *i.e.*, two immiscible liquids. His work is based on the different interfacial tensions existing, and his experiments and those of Hofmann, as mentioned in an earlier paragraph, have considerable bearing on the flotation problem.

Emulsions

Emulsions are fairly coarse dispersions of one liquid in another with which it is immiscible. The simplest and commonest emulsions are the oil-water emulsions, *i.e.*, the pure oil-water emulsions, containing no emulsifying agent such as soap, proteids, etc. In such systems the oil globules can be coagulated by electrolytes, show the Brownian movement strikingly, and can even be retained by some filtering media. Any process of emulsification is dependent on a surface tension lowering, or, to be more precise, on a lowering of the interfacial tension between the two phases. According to Briggs and Schmidt,[9] the two essential requirements of an emulsifying agent are these: (1) The property of condensing by adsorption in the dineric interface; and (2), the ability to form under these circumstances a strong coherent film. Temperature is a cogent factor in emulsification, for its effect is to reduce the interfacial tension between phases and also to lower the viscosity of the phases. In the production of emulsions, a considerable amount of surface energy is produced because of the relatively large surface area of the disperse phase; an emulsion is the more speedily effected if such surface energy be reduced by the use of a liquid having a low surface tension as the continuous phase. Some emulsions, under certain conditions, display a great increase in viscosity over that of either of the immiscible phases, *e.g.*, the emulsions of the Pickering order—up to 99 per cent. of oil in 1 per cent. of soap solution—can be cut into cubes. Any emulsion produced with soap solution is at once rendered *nil* by the addition of acid, as the latter will decompose the soap.

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[8] Ralston: *Mining and Scientific Press*, vol. 111, No. 17, p. 623 (Oct. 23, 1915).

[9] Briggs and Schmidt: *Journal of Physical Chemistry*, vol. 19, No. 6, p. 479 (June, 1915).

If solid particles are suspended in a liquid, they tend to increase the viscosity of that liquid only gradually depending on the amount of solid particles present. Experiments have shown that whenever a substance which is in suspension is wetted by two immiscible liquids simultaneously, it will go into the dineric interface in the manner already referred to, and will therefore tend to produce an emulsion. If, however, the suspended particles can not coalesce due to adsorbed oil film or for other reasons and thus effect the production of a coherent film, the emulsion will not be stable. Very little data are available on the production of emulsions by the oils used in flotation work, or on the matter of interfacial tensions between such oils and water. However, we are no doubt dealing with emulsified or partially emulsified pulp in some of the flotation processes, *e.g.*, the oil-froth process at least.

Electrolytic and Electrostatic Phenomena

Any substance which is placed in contact with water or many other liquids will assume an electric charge, the origin of which is, as yet, not set forth. Most substances when in contact with water become negatively charged, but these charges can be differed at will or reversed by the addition of the proper electrolyte in requisite amount. These electric charges are by no means confined to submicroscopic particles but are found also on the particles of a coarse suspension. Gangue minerals, *e.g.*, quartz, when suspended in water, are negatively charged, and sulphide minerals, *e.g.*, pyrite, are positively charged under like conditions. Oil drops are negatively charged, as are air bubbles under certain conditions which will be given subsequently. These charges are very minute when referred to the mass of the particle. Substantial evidence is at hand which goes to show that floatable minerals have the positive sign of electricity when suspended in water or can be made to assume that sign by the addition of proper electrolytes in sufficient amount. As Callow[10] observes, there is a parallelism between electrostatic characteristics and the flotation properties of ores. Many of the electrostatic principles have either been carried too far or misapplied as recent work shows.

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[10] J. M. Callow: *Bulletin* No. 108, p. 2334 (December, 1915).

Experiments in the realm of colloid chemistry indicate that the contact films are charged and that such charges markedly affect the dispersion or coherence of the particles in suspension. Naturally, oppositely charged contact films will coalesce while similarly charged contact films will repel each other, if the charges are sufficient in amount to overcome the force of cohesiveness; in the latter, dispersion is the result. The oil and air contact films having negative charges would tend to attract the sulphide particles, but further than this possibility electrostatics probably plays little part in flotation for the reasons given in the following paragraph.

As referred to above, it is pretty generally admitted that only minerals which are good conductors are suitable to flotation. Now then, as the electrical theory contends, electrified bubbles must be supplied to float the conducting minerals which are attracted, leaving behind the minerals which are not. The bubbles in flotation are simply air spaces contained by a mantle of oil or of water, and there is, therefore, nothing within to bear the charge. In case it could carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film. Further, electrostatics plays probably little part in holding the sulphide particles and the gas bubbles together as neither the bubble nor the particle can have a charge of sufficient magnitude when referred to the size. The electrical theory has been strongly championed by at least one writer[11] and has been tolerated by some others. A recent article[12] indicates that the principles of electrostatics have been considerably misapplied. It is my belief that electrostatics may be a small contributing factor in flotation in a manner not as yet understood because of a lack of data concerning charges at the interfacial boundary between immiscible phases, *e.g.*, where the colloidal state is introduced in oil-water emulsions. Apparently, as far as any data at hand are concerned, they all serve to condemn any attachment of great importance to the electrical theory.

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[11] Bains: The Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, No. 22, p. 824 (Nov. 27, 1915).

Bains: The Electrical Theory of Flotation, II, *ibid.*, vol. 111, No. 24, p. 883 (Dec. 11, 1915).

[12] Fahrenwald: The Electrostatics of Flotation, *ibid.*, vol. 111, No. 11, p. 375 (March 11, 1916).

THE CHARACTER AND FORMATION OF FLOTATION FROTH

Bearing in mind the phenomena discussed above, let us turn to the flotation of minerals by means of the froth process and look into the character and formation of froth. A froth has been defined as a multiplicity of bubbles; this seems to be wanting in some respects but will be requisite and sufficient for this purpose. The froth of flotation is formed by the action of air or gas in water containing impurities or contaminants, for pure water will not maintain a froth.

Froth and Bubbles

The idea has been abandoned of late, by most people, that a low surface tension is the essential requirement for froth formation. As mentioned by Coghill in a recent writing, the contamination of the liquid with an impurity which will cause a variable surface tension is the real requirement. A bubble of air is spherical in shape and this shape can only be maintained if the external pressure exceeds the internal pressure. Since a bubble does not expand *per se*, large bubbles can only be accounted for by heat, coalescence or electrification. Viscosity is an important factor in froth persistence as it increases the tenacity of the liquid film and thus prevents ready rupture. The rupture or bursting of bubbles is accounted for by these reasons:

1. Concussion upon a surface film which is deficient in the requisite viscosity and variable surface tension.
2. Relief of pressure—here the gas of the bubble in expanding exerts a pressure which exceeds that of the liquid film.
3. Adhesive force of the entrained gas for the atmospheric air.
4. Evaporation of the liquid film.

Flotation bubbles will burst then for any one or a combination of all these reasons.

Solutions in which the continuous phase is a solution of soap, various products from the saponification of albumens, etc., will froth voluminously even in a very diluted condition; frothing never occurs in pure liquids and is a definite proof that the solute or disperse phase lowers the surface tension of the solvent. A froth, which shows adsorption at the interfacial boundary of solution and gas, depends for its persistence on the production of a viscous film at that boundary; these viscous films are the direct result of surface adsorption of the disperse phase, *i.e.*, dissolved contaminants, the amount of which is small—disappearingly so. The work of Hall and of Miss Benson shows that in a foaming liquid the foam is richer in dissolved contaminant than is the bulk of the liquid. Froth formation in the Callow cell is the result of the injection of air into the pulp (already emulsified); the froth continues to obtain as long as there is sufficient air injected into pulp of the proper consistency. The nature of the froth in the Callow cell is governed by the kind of oil used and by the amount of air. A pneumatic froth is unstable or ephemeral, *i.e.*, it dies rapidly when removed from the influence of the injected air. The mechanical froth, on the other hand, is thick and persistent, and must be broken up in dewatering the concentrates.

Oils

Oils have a well-marked selective action for metallic sulphides, tellurides and some other minerals. The fact that both the oil and the air or other gas have a selective adhesion for sulphides prevents them from being wetted by water. Conversely, the quartz and other minerals exhibit just the opposite characteristics. The gangue minerals, generally, do not exhibit adhesion for either gas or oil and hence they are readily wetted by water. Gases have a well-defined adhesiveness for oils; hence the air or gas adheres strongly to the oil film. The stability of a froth depends, in the main, on the kind of oil used, *e.g.*, pine oil makes a weak brittle froth, and creosote makes a stable elastic froth. The work of Devaux[13] on oil films explains how so small an amount of oil as is used in the various flotation processes can be so efficacious. From a consideration of the immiscible oil-water interface, if any oil will film the internal surface of a gas bubble the sulphide mineral particles would be contained in the oil-water interface no matter what the nature of the gas contained by the water film. The sulphide if it enters the oil phase would then present an oiled surface to the water phase. There are three conditions then: (1) The mineral enters the oil phase completely; or (2) the mineral enters the water phase completely; or (3) the mineral enters the oil-water interface.

Some experiments made to determine the nature of the frothing power, selective and collective action of different oils show some interesting results. I made some tests on a zinciferous slime from the Joplin area with different oils and the results obtained indicate that a definite mixture of oils will effect better recoveries than any one oil alone. The best combination consisted of pine oil as a frother, plus wood creosote as a frother and selector, plus refined tar oil as a froth stiffener. The complete results of these experiments will be given in a later paper.

In general, pine oil makes a brittle froth which immediately dies; creosotes make a more elastic froth, the bubbles of which may expand to 3 in. in diameter or more before rupture. Coal-tar products are rather poor frothing agents and if used must be aided by either creosote or pine oil to produce a good froth. Oils of a lubricating nature seem to be of little value as flotation oils, while such light oils as gasoline and naphtha are of value only for thinning the heavy coal and wood tars.

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[13] Devaux: Oil Films on Water and on Mercury, *Smithsonian Report of* 1913, p. 261.

Air and Gas

At this time, there are three ways by which a gas may be forced into a solution mechanically, viz.:

1. By beating it into the solution by means of beaters or paddles—as in the Minerals Separation and similarly mechanically agitated machines.
2. By pneumatic means—as in the Callow cell where the air is divided by the porous blanket bottom into minute sprays.
3. By so-called liquid jets—as in a process recently patented in which the air is introduced as a surface film surrounding a liquid jet by surface tension.

Conversely, there are three methods by which dissolved gas may be expelled from a liquid, viz.:

1. When the liquid is supersaturated, the excess gas is expelled.
2. By heating the liquid, when some of the gas is expelled due to an increase in its volume.
3. By pressure reduction, as in the Elmore vacuum process, where, according to the law of Henry,[14] "the amount of gas dissolved by a liquid is proportional to the pressure to which the gas is subjected."

An air or gas bubble on being introduced into a liquid is at once surrounded by a film of the liquid. Such a bubble will rise to the surface (carrying the metallic sulphides by reason of the forces already mentioned) on account of gravitation, by which is understood that the adherence of the air to the liquid is less than the force of gravity.

RÉSUMÉ

From a consideration of the foregoing, it is believed that the theory based on the different interfacial tensions involved is the dominating one as well as the more consistent at this time. Probably flotation is due to a combination of phenomena which are rather high in the scale of complexity. The theory based solely on occlusion goes "by the board" as it has been shown that the contributing effect of this phenomenon has been interpreted rather laxly.[15] The phenomenon of electrostatics may be a small contributing factor but recent work indicates that the principles have been misapplied. An explanation more in consonance with fact can be given in terms of the interfacial tensions involved, without postulating either occlusion or electrostatics.

The main and essential requirements for froth flotation are: (1) The production of a persistent froth by any means; (2) the attachment of the bubbles of air to the sulphides or other material to be floated; and (3) the maintaining of a selective action of the froth bubbles for the sulphides or other material to be floated.

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[14] Jones: *Elements of Physical Chemistry*, p. 177, 1907.

[15] Ralston: Why Do Minerals Float? *Mining and Scientific Press*, vol. cxi, No. 17, p. 623 (Oct. 23, 1915).

Why this is accomplished is outlined in the above, particularly under the subhead, "Adsorption." Probably, before any generally satisfactory estimate of the many complex factors, which are supposed to be involved, can be secured, a more thorough investigation of all of them will have to be made than has thus far been attempted. However, in this line of inquiry there has been a steady progression of thought and a remarkable increase in knowledge in the past few years; and the art of flotation will continue to improve and develop as knowledge of the different factors involved increases and allows the exercise of a better control of them.

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The style is unhappily difficult, so that the following suggestions may not be amiss. After several readings of the short paper on The Theory of Colloid Chemistry, read first the excellent summaries at the ends of the Theory of Emulsification papers before undertaking a rapid survey of the whole set; follow this by a more careful consideration of the summaries with re-reading of portions of the text when necessary; continue this process until the drift of the argument begins to reveal itself.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Diastrophic Theory[*]

A Contribution to the Study of the Mechanics of Oil and Gas Accumulation in Commercial Deposits

BY MARCEL R. DALY,[†] SEATTLE, WASHINGTON

(Arizona Meeting, September, 1916.)

THE writer has devoted a number of years to practical operations and to the study of geology in the oil fields. In consequence, he has been brought to investigate the theories advanced to account for the accumulation of oil and gas in commercial deposits. It is the result of these investigations and his personal conclusions that he wishes to sum up in this paper.

The writer is an advocate of the organic origin of petroleum found in pools. He has been led to believe that the present theories of oil and gas accumulations are incomplete and, in certain respects, incorrect, although they may embody certain elements of truth; that the forces that are called upon to explain the movement are only secondary forces in the process, and insufficient, by themselves, to cause this movement; and that the phenomenon of accumulation is of much larger order than heretofore admitted and bears an intimate relation with the general as well as with the local deformations of the crust and is a direct consequence and a mechanical effect of these deformations to which the term "diastrophic" has been applied. As a complement to this theory, the writer offers a new interpretation of the "rock pressure" and of the mechanism of the "sealing up" of this pressure in oil and gas pools.

The Anticlinal Theory

The theory most generally accepted to account for the accumulation of hydrocarbons in commercial deposits is the anticlinal or structural theory. This theory, the principles of which have been laid down by I. C. White, Orton, Winchell and others, and which has since been modified and perfected, is too well known to need a detailed statement. A clear interpretation of it may be found, in the paper of W. T. Griswold, in *Bulletin No.* 318 *of the United States Geological Survey*, and the reader is referred to it. In this theory, the *force* which is supposed to have caused the motion is the gravity of the hydrocarbons; and the principal *factors* which have intervened in the process are the structure and the general stratigraphic conditions of the rocks, their texture and porosity, and their water content. The oil and the gas are supposed to enter the porous rock that is to act as a reservoir, by some indefinite process. Any further movement toward accumulation is only considered as possible if a dip exists in this rock sufficient to overcome the friction, and practically impervious areas of the rock will be more or less perfect barriers against the movement. Then, if this rock is completely saturated with water, or if the hydrocarbons have entered it below the line of complete saturation, the oil and the gas will gradually move up the slope by the effect of buoyancy; the gas, with its lower specific gravity, occupying the higher places. Should the rock be dry, or if the hydrocarbons have entered the rock above the line of saturation, the oil will flow down, as long as gravity is sufficient to overcome the resistance to motion, and the gas will diffuse with the air or water vapor contained in the pores of the rock.

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[*] Originally presented at a meeting of the Puget Sound Section on Jan. 29, 1916.

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From this statement it will appear that the following assumptions are an absolute requisite for the anticlinal theory:

First, the structural deformations (dips, anticlines and synclines, domes, terraces, etc.) must be supposed to have existed previous to the introduction of the hydrocarbons in the porous strata; in other words, it must be assumed that structural deformation has preceded the movement toward accumulation. Indeed, in this theory, the preëxistence of structural deformation is the very basis of the action of gravity and, at the same time, the theory implies that the forces which have caused the deformations have had no bearing on the movement of the hydrocarbons.

Second, gravity or buoyancy is to be considered the sole agency through which accumulation has been brought about and, as such, is supposed to be adequate to explain accumulation under any condition of dip.

To which the following considerations may be added: The movement toward accumulation would have to take place in the porous rocks when they are already solidified and partially cemented. There would be no connection between the causes which have led: To the introduction of the oil in the porous strata; to the movement of the fluids in these same strata; and to the "rock pressure" itself. Further, this theory does not provide for any satisfactory explanation as to the mechanism of the "sealing up" of the pressure in the pools.

The first assumption is begging the question. To the best knowledge of the writer, no attempt has been made by the authors of the theory even to discuss the point or to prove the accuracy of the inference. It seems as if, from the start, it was admitted as self-evident that the gravity of the fluids was the only possible force entering into action. The writer will try to show later on why, in his opinion, this assumption falls short of truth and how its admittance may be explained by the fact that the problem has not been attacked from its true angle. On this first point, to say the least, the anticlinal theory is not established on proven ground.

The second assumption may be seriously contested. The force of motion due to buoyancy is a function of the sine of the dip. It would be maximum for a vertical stratum and null for a horizontal one. Many anticlines in the Eastern fields of the United States have a dip so low that a motion caused by buoyancy alone could hardly be understood. Dr. Ashburner has found a maximum dip in the Bradford region of 69 ft. per mile, or less than 1 ft. for 76 ft., while Carll has shown that the dip of the oil sands in the Venango belt, and the southern end of the Butler belt, rarely exceeds 34 ft. per mile (Sir Bowerton Redwood), or less than 1 ft. for 155 ft. In West Virginia, the maximum dip of many anticlines is less than 1 ft. for 50 ft., which corresponds to an angle of about 1° 8′ 45″. For this last dip, the force due to buoyancy, which is supposed to move the oil up the slope, would be reduced to 0.0066 of the weight of the oil, as pointed out by Malcolm J. Munn, of the U. S. Geological Survey. A volume of 1 c.c. of oil would thus be submitted to a moving force of a few milligrams. The oil could be set in motion only if this force were greater than the resistance due to static friction, viscosity and surface tension. It is not possible, with the data now at hand, to submit the question to a complete mathematical treatment. As Van Hise points out,[1] "the pore openings in sandstones are for the most part capillary", and the flow of oil through the rock would have to obey the laws of flow through capillary tubes, which are not yet satisfactorily expressed. Further, the pore openings would be extremely irregular in shape and dimensions. But it is possible to reduce the problem to a simpler form. If, barring viscosity and surface tension, we would only consider the resistance due to static friction, we would obtain an inferior limit to the resistance. And if it were possible to show that the force of motion due to buoyancy is insufficient to overcome such a frictional resistance, in the case under consideration, it would be proof that the anticlinal theory can not be accepted, at least in its present form, and that other forces besides buoyancy are to be considered in order to explain the motion.

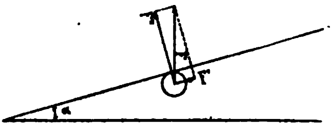

Fig. 1.

Let us imagine a globule of oil appressed against the roof of the sandstone stratum by the effect of buoyancy (Fig. 1). Let F be the force of buoyancy. This force may be decomposed into two components: F' parallel to the dip, which would cause the globule to slide along the roof, and F'' normal to the roof, which would appress the drop of oil against it.

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[1] A Treatise on Metamorphism, *Monograph* 47, *U. S. Geological Survey*, p. 138 (1904).

The effect of this pressure is to create a resistance to the sliding, which is proportional to the pressure and directed down dip. This proportion, which remains constant when the nature of the surface remains the same, is the friction coefficient. Let it be c. The movement will become possible only if $c < \frac{F'}{F''}$ or $c <$ tang α, α being the angle of the dip. If the dip is 1 ft. to 50 ft., the tangent is about 0.02 and the condition reads $c < 0.02$.

There are no data available for the friction coefficient of oil and sandstone under the conditions stated. But one may get an idea of the order of magnitude of this coefficient by a simple experiment. Let us take a slab of some oil-bearing sandstone, some Berea grit, for instance. The lower surface of this slab must be planed, but not ground, so that the grains of sand remain entire. Let us immerse the slab in some salt water of proper density, in a horizontal position, and loosen a drop of oil from a syringe in the water, under the slab. The drop will rise to the bottom of it, and here it will remain if the slab is perfectly horizontal. Let us then incline the slab progressively and note the smallest angle of dip at which the drop will be set in motion. The tangent of this angle will be equal to the coefficient of static friction.

The writer has made this experiment repeatedly and has never succeeded in getting the drop in motion for any such angle as 2°, which would tend to prove that the coefficient is superior to the maximum limit required.

It is further to be noted that the spherical form is the general form of oil drops in an emulsion, where the particles of oil are, in great majority, very small; it is also the form which, every other thing being equal, offers the least resistance to motion. Hence, if the motion is shown to be impossible under this form, it will be equally impossible under any other form, and the whole theory falls through.

The change from a spherical form to an irregular one, by the flattening of the drops when compressed against the walls or when passing through irregular and narrow channels, the indentations, the penetration of the oil between the grains, the effect of viscosity, the capillary action—especially if gases be held in the emulsion—would be as many causes furthering an increased resistance to motion.

Now, the fact that the movement toward accumulation is not possible under the sole influence of gravity, with the low dips so characteristic of many oil regions, does not mean that gravity has had no influence in the process. It simply means that this force alone would be inadequate to explain the motion and that some other force or forces must have entered into play. But though gravity of hydrocarbons may not be a primary agent of their accumulation, it is a force which must be taken into account in any general theory of the process.

The Hydraulic Theory

In opposition to the anticlinal theory of oil and gas accumulation through gravity, Malcolm J. Munn has propounded[2] another theory in which the moving force would be the hydraulic (not hydrostatic) pressure and capillarity of underground waters. This is termed *the hydraulic theory.*

"The fundamental idea of the hydraulic theory," writes Mr. Munn, "is that moving water under either hydraulic or capillary pressure has been the direct agent of accumulation of oil and gas pools. To this idea may be added another of equal value—the pools of oil and gas are held in place by water under hydraulic and capillary pressure which effectively seals up all the pores of the surrounding rock and prevents the dissipation of pressure by diffusion." This theory is very interesting and, as will be seen later on, the writer agrees with Mr. Munn on two points, viz., that water under hydraulic pressure has really been the primary agent of motion, and that the "sealing up" of the pools is a phenomenon of surface tension. But, as far as the writer can see, Mr. Munn has reduced the hydraulic pressure to that of underground waters circulating in the same way as they do now, and his statement relating to "hydraulic and capillary pressure" is rather undetermined.

"Capillary pressure" is a somewhat misleading term. If by this is meant the general action of forces due to surface tension, it is well to note that surface tension may create a resistance against the flow as well as a tendency toward it, as Mr. Munn himself seems to acknowledge when he comes to the "sealing up" of the pools. If it simply means the process of infiltration, the creeping of a liquid through capillary channels, this process has its limitations and would seem to be a process of dissemination rather than of concentration. On the other hand, hydraulic pressure of underground waters may be inadequate. Hydraulic pressure is a function of the square of velocity, and the velocity of underground waters is known to be extremely small. "The motion of the ground water as a whole," writes C. S. Slichter,[3] "is somewhat like the slow motion of very viscous sirup or the slowly creeping ice of a glacier." It has been a shown, for instance, that the ground waters of the Arkansas River flow in gravels at a rate not greater than 3 to 5 m. a day. This would mean a velocity between 0.00361 and 0.0058 cm. per second, or less than from 13 to 21 cm. per hour. It is possible to show that, should the static coefficient of friction between oil and sandstone be greater than 0.02, as heretofore admitted, a velocity of 0.006 cm.-sec. would not overcome the frictional resistance of oil in an inclosed pipe, on a horizontal plane, barring the additional resistances due to viscosity, surface tension, deformation, etc., which would interfere in this case. Though the analogy may not be entirely correct, it has a tendency to show that the influence of hydraulic pressure of underground waters is here doubtful and at least remote in importance. There will be always a great difference between the transportation of a material in the state of solution, which may be followed by subsequent re-deposition, and the transportation of insoluble solid material, as oil in water. Further the theory is open to the same general line of objections as the anticlinal theory, and more so, as structural deformation, which is so intimately connected with the phenomenon of accumulation, becomes here entirely subordinate and incidental.

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[2] *Economic Geology*, vol. 4, No. 6, p. 523 (September–October, 1909).

[3] The Motions of Underground Waters. *Water Supply. Paper, No.* 67, *U. S. Geological Survey*, p. 35.

The Diastrophic Theory

The fundamental weakness of the theories which we have reviewed may perhaps be traced to the hesitation of the authors of these theories to choose deliberately between the different conceptions of the origin of oil itself. This origin once ascertained, it is evident that the surest way to determine how oil and gas may have accumulated in certain places, would be to try to follow the possible movements of the oil from the time of its first appearance in the strata, down to the pools where we find oil today. In order to be rational, a history ought to be complete; and we have no more right to limit the history of petroleum to the space of time during which the strata, in which we find it now, may have existed in the same present shape and condition, than we would have the right to reduce the story of a man to the last years of his life. It is further obvious that the history of petroleum will have to be entirely different, dependent upon its origin, either from emanations coming from the depths, or from organic decomposition in the strata themselves. In the writer's opinion, many reasons favor the theory of organic origin for the petroleum found in commercial deposits, though this does not mean that petroleum of a different origin does not exist elsewhere. Hence, this is a point on which we have first to agree, or else agree to disagree. Should the organic origin of the petroleum that is found in pools be granted, the following interpretation is offered for the mechanics of its accumulation.

Oil developed by the process of decomposition of organic matter—whether vegetal, animal or both—would have first to exist in the sediments which contain the parent matter, in the state of dissemination. As Orton remarks, "disseminated petroleum is well nigh universal, but accumulations are rare."

In order that such an accumulation of disseminated particles may take place, three elements are necessary: An adequate source of supply; a reservoir to hold the oil and accompanying gas under pressure, and a process of concentrating the disseminated material and conveying it to the reservoir. The source of supply will have to be ascertained in every individual case. It is supposed here to exist, for the sake of argument. The reservoir is easy to imagine, inasmuch as we are able to corroborate our theoretical views by actual knowledge. A porous stratum overlaid by an impervious one, may act as a reservoir. This is the general case. Or the reservoir may be constituted by a porous stratum highly inclined and cut off abruptly below ground by a fault, which thus seals the oil and gas and prevents their escape. This kind is found in some Californian fields and elsewhere. Or the reservoir may be constituted by joint cracks in a shale, as is the case in the Florence oil field, Colorado,[4] etc.

The interpretation of the process of concentration and eventual migration, *i.e.*, of accumulation, is more complicated. As a natural process, it must obey physical laws, and especially the laws which govern the motion of fluids and gases; but the *forces* which may cause the motion (gravity in its different forms, heat, surface tension), the *agents* through which these forces accomplish certain results (water, oil and gas in varying physical or chemical states), and the *factors* that intervene in the process (structure and texture of the rocks, porosity, stratigraphic conditions, water and gaseous contents, viscosity, capillarity, depth and time) may have or have had a widely varying range of influence, according to past and present conditions. These elements would have to be seriated, according to their probable rank of importance, and their relative degree of influence determined, before a tentative conclusion could be reached. It thus becomes evident that if some broad general principle may be laid down, a great number of variations are to be expected for each individual case.

A few general observations may help us from the start. According to the organic theory, which is accepted here, the ultimate source of the hydrocarbons is to be found in sedimentary organic deposits, and oil accumulations in commercial quantities (oil pools) are always associated with sedimentary strata. The immediate inference is that hydrocarbons must have been submitted, from the date of their origin, to the action of the forces that may have affected these same strata.

If we refer to a map of the known oil fields of the globe, we will notice, with Sir Bowerton Redwood, that "whilst petroleum exists very generally distributed throughout the world, the principal deposits occur along well-defined lines, often associated with the principal mountain chains." This remark is important and deserves to be emphasized. It means that petroleum deposits are mainly associated with the lines of lesser resistance of the globe, with the general direction of geosynclines and consequently with the areas of general orographic movement and deformation. The distribution of the Eastern oil fields of the United States and Canada along the Paleozoic trough, or geosyncline, of the

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[4] C. W. Washburn. *Bulletin No.* 381, *U. S. Geological Survey.*

Appalachian region, the location of the Western oil fields of the Pacific coast, from Alaska to California, followed by the Mexican oil fields, the deposits of the West Indies, of the northern coast of Venezuela, Columbia and a part of the Andes, along the general trend of a Mesozoic syncline; the oil fields of Galicia, Roumania, of the Caucasus, of Burma, of the Islands of Java, Sumatra, Borneo and New Zealand, following the path of another syncline, are extremely suggestive.

This first observation may be completed by a second. An oil field is always accompanied by a certain amount of local structural deformation, which sometimes is reduced to simple undulations, as is the case of the Appalachian region, or may reach a stage of high disturbance, with contorted strata, as is the case of the Galician oil fields. But local deformation has been observed everywhere. It becomes thus difficult to escape the impression that structural deformation, general as well as local, is more or less connected with the phenomenon of accumulation, and our inquiry is thus directed toward the possible action of the forces which have produced these deformations in the sediments from which hydrocarbons have proceeded and their possible action on the hydrocarbons themselves.

As stated before, oil would have first to exist in the stratum where organic decomposition takes place, in the shape of a finely disseminated matter. The sediments themselves would have to be deposited under water, in lagoons, marshes, deltas or at the bottom of the seas, in the form of mud or ooze. Such a mud would be composed of extremely fine mineral particles, intermingled with water in large amount. The particles of oil would take the form of spherical globules under the influence of surface tension, and, as these globules would be larger than the mineral particles of mud, they would be mechanically held in the mixture. Experiments conducted by Murray Stuart, Assistant Superintendent of the Geological Survey of India and others, have clearly established this capacity of muds to hold oil in sediments by purely mechanical action.[5]

The preceding condition may be termed the *first phase* of the process. It is characterized by a layer of mud or ooze, holding mechanically, between its very fine mineral particles, disseminated drops of oil and gas bubbles, in a "matrix of water," as Murray Stuart expresses it.

When the conditions of sedimentation which allowed the deposition of the mud have changed, some other material may be deposited over this first layer; say some sand. Then, another change in sedimentary conditions will occur, and a new layer of material will cover the sand; suppose another layer of mud; and so on, successive layers of various sediments will be piled up in the sedimentary syncline.

Now, the progressive increase of weight due to this accumulation of successive layers will cause a progressive compression to take place. The effect of compression on a mass of practically incompressible mineral particles imbedded in water is to draw these particles closer together, to reduce the space available between them, and to squeeze out more or less of the liquid contents. The limit of compression is reached when the particles come in contact. For this reason, the result would be somewhat different, according to the size of the component particles of the layers. Mud or ooze particles are exceedingly small. When, in the syncline, the lower layer of mud would be compressed, the resulting texture would be close, and the tendency would be for the fluids to be squeezed out to a great extent; whereas the grains of a layer of sand are much larger than the particles of clay, and such a layer may become incompressible when it still contains a great percentage of holes. It may thus act as a reservoir for liquids even under considerable pressure. The overlying sand stratum would then remain porous to a large extent, and the liquids, escaping from the underlying mud, would rise and become confined in it. This may be termed the *second phase* of the process, and is characterized by the progressive transfer of a large part of the oil and gas from the clayey layer in which they have originated to the overlying porous stratum in which they still remain distributed in a finely disseminated state. At this stage, the porous rock may be supposed to be saturated with an emulsion of oil and gas in water.

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[5] *Records of the Geological Survey of India*, vol. 40, Pt. 4, pp. 320–333 (November, 1910), and *Economic Geology*, vol. 7, No. 1, p. 91 (January, 1912).

Finally, in the course of time, sedimentation is stopped in the syncline of deposition, and orogenic movement begins. Whatever may be the cause of this movement, it is a tendency toward a new adjustment which evinces itself by a lateral thrust. In the simpler form, one rim of the syncline remains immovable, and acts as a resistance, and the other rim is brought nearer to the first one by a movement tangential to the crust, which squeezes the syncline as between the jaws of a vise. The strata are bent and lifted above the level of the waters.

It is possible to get an idea of the order of magnitude of the thrust and to compare it with the forces due to buoyancy and hydraulic pressure invoked in the preceding theories, and which we have found to be insufficient to explain the migration of the oil. For this, we have to consider that this new force exceeds the resistance to crushing offered by the most resistant sedimentary rocks. This crushing strength becomes thus an inferior limit to the force. Limestone, for instance, may be crushed under a pressure of between 400 to 1,200 kg. per square centimeter; so the thrust is to be measured at least in hundred kilograms to the square centimeter. We have already seen that the force of buoyancy, by which a cubic centimeter of oil is supposed to be lifted along a dip in the anticlinal theory, is to be measured in fractions of a gram (in milligrams for a dip of 1 ft. for 50 ft.), and hydraulic pressure due to underground waters, with the velocity admitted, would be still less important. The magnitude of the force we are considering now—the tangential thrust—is then at least one million times greater per unit surface than the two other forces previously considered. Further, this force has left its imprint in all sedimentary strata, not only by the flexure of the strata or their relative displacement, but frequently by dynamo-metamorphic actions which it has brought about. These actions have taken place between the time the strata were lying at the bottom of the waters and the time they have acquired their present situation. If then, any oil was ever existing at the time of the deposition of the strata and has remained in some stratum since, it must equally have been subjected to the action of the thrust. The *third phase* of the process begins with this action of the thrust. But this needs a more complete analysis of facts.

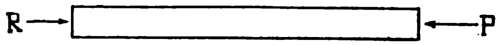

FIG. 2.

When a square prism of a solid material, such as a piece of wood or an iron bar (Fig. 2), disposed horizontally on a support in order to avoid the action of gravity, and propped against a resistance *R* at one end, is compressed at the other end by a force *P*, in the direction of its axis, the compression is instantaneously transmitted to the resistance, from end to end. Then, if the pressure is sufficiently increased, the prism will bend (Fig. 3). This flexion will extend the fibers on the convex side and contract them on the concave one, and there will be an intermediate layer which will be neither contracted nor extended by the flexion. This is the neutral plane (Fig. 4). Superimposed to this first effect, there will be a general contraction along the axis, which will affect the whole prism.

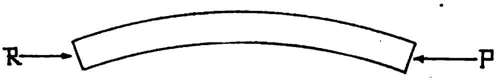

FIG. 3.

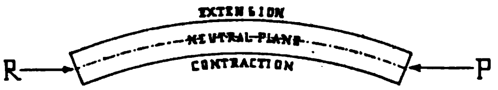

FIG. 4.

If, instead of a prism of solid material, we operate on a prism made of some homogeneous plastic material, like moist clay, the result would not be exactly the same. The compression at one end would not be immediately transmitted at the other end, but only progressively, in proportion as the inertia of the successive sections of the prism and the friction against the supporting stand would be overcome, the one after the other. And deformation would begin at the end where the thrust *P* is applied, even before any compression would become noticeable at the other end, if the prism were long enough.

Let us now imagine a prism composed of a series of parallel, superposed, horizontal and homogeneous layers, individually uniform in thickness, but variable in composition and resistance, such as clayey, sandy and calcareous layers would be. Suppose such a prism submitted to a uniformly distributed load, representing the load of superincumbent strata; the friction against the stand reduced by the interposition of a soft base, and proper precautions taken to avoid lateral deformation. And let us subject this prism to the same process of compression. Here again the compression would be transmitted progressively from the point of applied pressure to the point of applied resistance, and deformation would begin at the extremity nearer to the thrust. This deformation may vary, according to the relative thickness and disposition of the "competent" and "incompetent" strata, as they have been termed, but would generally take the form of a local bend, some sort of anticline, which may be overthrown in the direction of the resistance. If now the pressure should be continued, waves would rise in succession, from right to left, decreasing in height, the farther they reach toward the resistance, and the flexed prism would finally take the general appearance of Fig. 5. The mechanical result would be: (1) A general compression of the prism, which would decrease from *P* to *R*, where it might disappear, and (2) deformations by bending, with appressed folds toward *P* and lower anticlines toward *R*; the folds being generally overthrown in the direction of the resistance *R*.

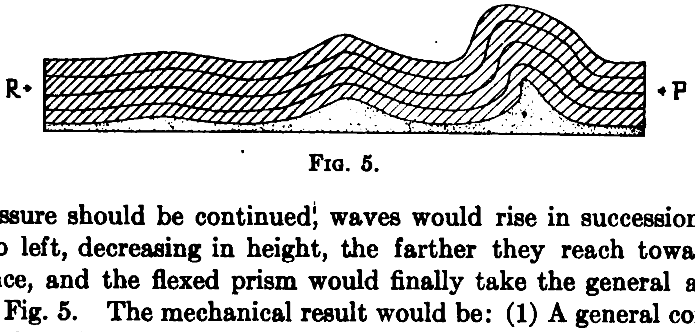

FIG. 5.

Now, we have already seen that when a material is compressible, compression will tend to draw nearer to each other the solid particles of which the material is composed. Compression would thus tend to a reduction of volume and to a reduction of porosity, or capacity for liquids. The first mechanical effect of our experiment, then, tends to reduce the capacity for liquids of the material at a decreasing rate from *P* to *R*, from the point of applied pressure to the point of applied resistance.

To this would be superimposed the mechanical effect due to flexure. Anticlines would be extended on their convex side and synclines compressed on their concave side. If, then, the thickness of the prism is sufficient, and if we consider only its upper portion, above the neutral plane, the general effect would be as follows: A general decrease of compression and a general increase of capacity for liquids from *P* to *R*, superimposed on a wavy succession of local increases and decreases of capacity. This would create a succession of zones of increasing capacity from right to left (from *P* to *R*), individually elongated in a transverse direction to the prism, or in a parallel direction to the folds. If then the porous strata were saturated with a liquid, the tendency would be for the liquid to move from the more compressed parts to the less compressed or decompressed ones. This movement would take place along the lines of lesser resistance; viz., (*a*) from the clayey layers to the sand strata, which would complete the concentration of the oil emulsion, already begun by the effect of vertical compression, into the sandy layers, and (*b*) along the sand strata themselves. Pools would be constituted parallel to the flexures, which would become more important the nearer they approached the resistance, or increasing from right to left.

Further, compression and consequent migration of the emulsion through the sand would have a tendency to allow oil drops and gas bubbles to coalesce. Temperature may play a part in the phenomenon, as pressure would generate heat. This heat may be negligible in the zone of reduced compression, but may be important in the highly compressed one. Rocks may afford sometimes a certain amount of heat without noticeable changes; but organic liquids would be more or less modified. A slight increase of temperature would lower the viscosity of the oil and facilitate its migration. A sufficient rise may partially decompose it, and complex reactions may occur if this decomposition should take place under pressure, in the presence of water vapor. This side of the question will have to be investigated more completely, should the general theory be admitted. For there could be found possibly a partial origin of the different grades of oil, and the source of a large part of the gas which accompanies oil in the pools today. The complexity of the products would still be increased by the filtration of the oil through certain rocks, which an excessive compression may allow, or which capillarity may induce in other circumstances.

At any rate, gas and oil concentration in the zones of decompression or of reduced compression would result; and water, oil, and gas may finally settle in the reservoirs according to density.

On the other hand, we have seen that the anticlines would be extended along their crests and the synclines contracted along their bottoms, if we consider only the part of the prism above its neutral plane. The liquid would thus be squeezed out from the synclines and would have a natural tendency to collect, by the effect of compression, in the higher and more open parts of the anticlines. Later, if, for some reason, water were reduced in amount, the oil would follow the level of the receding waters and sink toward the bottom of the synclines, as long as porosity would permit.

It might occur, with a sufficient increase of the lateral thrust, that the material subjected to compression would break instead of bend, and that overthrusts and faults would result. These deformations would naturally take place in the more compressed region, on the side of the thrust. This would open a way of escape to the gas and to the liquids, and the affected parts of the oil-bearing strata might be deprived of their contents, by outflow or evaporation. The oil and gas would be lost and these parts of the layers would become dry.

As far as we have gone, we have only considered theoretical views and results of laboratory experiments. The material which we have been handling was simple, the individual layers homogeneous, of constant thickness and horizontally disposed; the forces were acting in the vertical plane of symmetry of the prism, etc. And from a starting point fraught with geometric simplicity we have reached simple and geometric results. The merit of this way of proceeding is that it has allowed us to understand more clearly than could be done otherwise, the trend of events when the elements of the problem are reduced to their simpler lines.

But in Nature, things are much more complicated. Let us consider, instead of a prism, a syncline of deposition and note the new set of conditions which may affect the result. Between the geometric dispositions of our previous scheme and the dispositions encountered in the field, we will remark at once the following differences:

The material will not be homogeneous, but varied to the extreme. Not only will the strata be different from one another in composition, texture, resistance, flexibility, etc., creating a lack of homogeneity in the vertical direction, but similar differences will be found in individual strata, so that the lack of homogeneity will become almost general.

The materials will not be geometrically disposed. The outlines of the compressed area, in horizontal projection, will be irregular, and its thickness at great variance from point to point. The layers will be incurved, presenting original bends of deposition. Their own thickness will be irregular, and the so-called "planes of stratification" will not be planes at all, but complicated surfaces.

If we consider one particular stratum, the pressure due to overlying formations will not be the same everywhere.

The thrust will be irregular, and to simple compression and flexion will be superimposed a torsional movement. Consecutive thrusts, varied in direction and importance may affect the same region. Further, an orogenic movement does not result simply in a series of more or less parallel folds, elongated in one direction; it is complicated by transverse or orthogonal undulations, which create successive elevated and depressed areas along the general strike of the folds; the anticlines have a tendency to be transformed into elongated domes, the synclines into elongated basins.

located on the exterior slopes of the belt; *i.e.*, on the side that is farther from the point of application of the thrust, and the pools are distributed in rows parallel to the trend of the folds, frequently with their major axis in the same direction.

In these districts the deformations are no more the gentle anticlines and synclines or the terraces of the Appalachian region; the strata are highly disturbed, contorted and even faulted. The thrust in the Alpine belt has been much more powerful than in the Appalachian region, and in certain places overthrusts of considerable magnitude have taken place. In fact, the Alps proper and the Carpathians seem to have been submitted to folding and deformation at least at two different periods: First immediately after Oligocène times, and second during the middle Miocene; and the present state of affairs is the result of the cumulative effect of both thrusts. It seems reasonable to admit that the region has passed through a first stage similar in some respects to the one presented by the Appalachian belt, and during which the greater part of the accumulation may have been effected, before reaching the further stage of greater disturbance, in which a part only of the accumulated hydrocarbons seems to have been preserved, in consequence of special stratigraphic conditions.

Similar relations may be observed in many other fields.

The preceding remarks have led the writer to believe that the facts observed in the field agree with the theoretical views previously expressed, and the hypothesis which he proposes for oil and gas accumulation may be summarized as follows:

First Stage.—The oil proceeding from organic remains, perhaps still in process of decomposition at the very origin of the movement, is at first distributed in the water-laden sediments of the geosyncline of deposition in the state of disseminated particles.

Second Stage.—The increasing compression, due to the continuous accumulation of superimposed strata, expels an increasing amount of the water of deposition with its contents of hydrocarbons, from the original layers, which, at the same time, are the most easily affected by the compression (argillaceous or limy sediments), to some other layers less affected by it (especially sands and gravels). This displacement may take an upward or a downward trend, the only condition being that the fluids must move along the line of lesser resistance, from a more compressed to a less compressed zone. This movement may take place between two entirely different strata, or between two layers of the same group of deposits, provided the layer acting as a temporary reservoir is less compressible or more porous than the former one. (This latter condition obtains in the dolomitic layers of the Trenton rock.)

Third Stage.—As soon as orogenic movement begins, a more or less horizontal compression, due to the thrust, takes place and becomes added to the vertical pressure due to superincumbent weight. The result of the intervention of this new force is to create: first, a general increase of compression from the point of applied resistance to the point of applied lateral pressure; second, successive and parallel zones alternately compressed and decompressed, whose strike is normal to the direction of the thrust. The waters which saturate the strata are submitted to the effect of this unequal pressure and move from the highly compressed regions to the lesser compressed ones, carrying the hydrocarbons with them in their course, finally collecting in pools parallel to the folds. The movement would have to take place along the lines of lesser resistance, *i.e.*, toward and along the more porous layers of the formation (sandy layers, etc.). Pressure would reduce the viscosity of the oil, favor the coalescence of the globules and perhaps induce some chemical changes of the hydrocarbons. The more probable places of accumulation would be the crest of anticlines, the summit of domes, the rims of terraces, or, in the main, the places where a change occurs in the dip or along the strike of the strata, in the form of convex edges or arches; for the reason that at these places the local reduction of compression, buoyancy, and resistance to motion due to a change in the direction of flow, would act together and accumulate their effect.

When the fluids reach the zones of lesser compression, if the physical condition of the reservoir in which they collect is such as to hold the hydrocarbons and prevent their escape, an equilibrium is established, and the final pressure in the pools must be equal to the original pressure less the losses of head encountered on the way.

Further, a progressive settlement would take place in the reservoir, according to gravity; water would congregate at the lower places, oil would have a tendency to collect at its surface, and gas—either brought with the water or dissolved in the oil and further released by decompression, or simply produced from the oil itself—would reach the higher places.

Considered as a whole, the process would be a consequence of the mechanical principle of least action. In this process, the agent of transportation of the hydrocarbons would be water; the moving force would be hydraulic pressure created by vertical and lateral compression; and the extent of the movement would be variable in the extreme according to local conditions of the strata. But compression, especially compression due to the lateral thrust, whose action would be irreglar and continuous and would have to be extended over a long period of time, would not act as the permanent head of water which nowadays is the ultimate source of the velocity and of the hydraulic power of circulating underground waters. Frictional resistance may reduce and even stop the motion of underground water once for all in a given direction; a thrust would act by a succession of jerks and repeat the effort again and again. There would be periods of activity, during which the compressive force would exceed the resistance of the rocks and deformations would ensue, followed by periods of rest, brought by the momentary relaxation of strains due to deformation itself and during which the compressive forces would accumulate; and the thrust would become the source of a periodically renewed energy. The consequent hydraulic pressure would follow a similar wavy movement, with periods of maxima, to which a maximum of velocity would correspond, and periods of minima or of rest, where frictional resistance might bring the movement to a stop. The action of the liquids would thus become similar to that of a water ram, with a maximum of efficiency periodically renewed.

Fourth Stage.—A new stage will be reached by the gradual reduction of the water contents of the strata, producing consequent changes in the level of complete saturation and in the local disposition of the pools, by gravity.

Fifth Stage.—Sometimes, a new period of folding may take place, in which the thrust may have or may not have the same direction as the previous one. New zones of compression and decompression may be created, and the liquids may be put again in motion. The results may become thus very intricate, especially if the strata are deformed to a large extent.

All gradations must be expected to be found in the oil fields between these two extreme sets of conditions. The Appalachian belt may represent the first set, which stops at the fourth stage of our description. The fields of central Europe would represent the further and more complex stage.

The research for oil becomes thus a problem of tectonic effects as well as of stratigraphy.

Rock Pressure

The origin of "rock pressure" has been traced to one of the following causes: Hydrostatic pressure, weight of superincumbent strata, gradual accumulation of the inclosed gas, capillary diffusion.

The hypothesis of hydrostatic pressure is untenable as a general one. Hydrostatic pressure would agree, to some extent, with the pressure found in Ohio and Indiana, but could not account for the heavy pressures encountered in western Pennsylvania, as shown by Prof. J. P. Lesley and J. F. Carll, nor for the pressure of most of the deep West Virginia wells. Further, this theory is contradicted by the well-known fact that flow and pressure are found to decrease in any given well with the age of the well. A certain amount of constancy in the flow and in the pressure ought to be expected under artesian action, which is not the case.

The weight of overlying strata, under *present* conditions of the rocks would be mechanically inadequate.

The progressive accumulation of the gas may be a cause of pressure, and, according to David T. Day's experiments, capillary diffusion through

Fuller's earth bears a curious analogy with osmotic phenomena and the pressure due to this cause may be compared with osmotic pressure. But, if both theories may explain the origin of a certain pressure, they are not entirely satisfactory, in the writer's mind, for the following reasons: The progressive accumulation of the gas would rapidly be checked by the increasing pressure itself, and it has not been shown that such a limitation would not occur before reaching the high pressures encountered in some wells. On the other hand, the action of capillary diffusion seems to require certain physical conditions which are not met everywhere, and therefore it can not be admitted as a general cause. Further, none of the preceding theories explain the following facts: At least in the eastern fields of the United States, the rock pressure, in the main, increases with the depth of the "sand," and, at the same time, there seems to be a decrease of pressure with an increase of distance from the principal axis of the folding. The closed pressure in the Trenton limestone of Ohio and Indiana averages 200 to 300 lb. per square inch and only exceptionally reaches over 600 lb.; whereas the pressures in western Pennsylvania and in West Virginia, farther east, easily reach the double figure. In other words, it seems as if the pressure would increase, as a whole, in the same direction as the compression to which the rocks have been submitted at the time of their folding, both vertically and horizontally. A partial origin of the rock pressure, at least, would thus have to be traced to orogenic deformation. The two other forces—due to progressive gas increase and capillary diffusion—may have, and possibly have played a more or less important part in the final result, but this effect would have been produced later, and, in this respect, is to be considered as subordinate.

How Capillary Pressure Seals an Oil Pool

One of the most interesting problems involved in the study of oil and gas accumulation is the process by which gas or oil may accumulate in the pay streaks under heavy pressure, without this pressure being dissipated through neighboring rocks or through the sandstones. Imperviousness of the superincumbent strata and of the oil-bearing bed itself has been frequently advocated. But unaltered rocks of the type encountered in oil fields are never impervious. From 10 to 40 per cent. of their bulk is made of pore space, and the pore space of the inclosing beds of shales averages no less than 6 per cent., which means that every square foot of so-called impervious rocks contains an average of 8½ in. of holes (M. J. Munn).

In his "hydraulic theory," M. J. Munn has suggested[6] that "the pools of oil and gas are held in place by water under hydraulic and capillary pressure which effectively seals up the pores of the surrounding rock, and prevents the dissipation of pressure by diffusion." "Pressure in pools," he writes, "is maintained by the expansive force exerted by gas. . . . Such gas could not diffuse because of the saturated conditions of the surrounding rocks." But Mr. Munn does not go further than a general statement, and no detailed explanation of the process by which such a "sealing up" is rendered possible has yet been given, as far as the writer knows. The following is proposed as a tentative explanation.

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[6] The Anticlinal and Hydraulic Theories of Oil and Gas Accumulation, *Economic Geology*, vol. 4, No. 6, pp. 523 and 527.

Hydraulic pressure is here discarded, this problem being a problem of statics rather than of dynamics, and in which the velocity is naught.

There is a peculiar and interesting phenomenon which appears to have escaped the notice of those who have tried to explain the "sealing up" of a pool. It has been pointed out by Jamin, a French physicist, and is illustrated by the experiment of "Jamin's tubes." If a capillary tube is incompletely filled by water and the water distributed through the tube in such a way as to constitute a string of droplets, a pressure may be applied to one of the extremities of the tube which will not be transmitted to the other end. In other words, the string of droplets will act as a resistance. If there is a large number of drops in the tube, the difference of pressure at the ends arising in this way may amount to several atmospheres. The explanation follows: A drop of liquid, like water or oil, which does not wet the tube, will be limited on both ends by a meniscus (Fig. 7). The superficial tension which results in this form is caused by the tension of glass and air, glass and liquid, and air and liquid. The two first sets of forces are parallel to the axis of the tube, and, being equal and directly opposed two by two at both ends of the drop, neutralize each other when conditions of equilibrium are formulated. The third set of forces, caused by the tension of air and liquid, is tangent to the meniscus all around the tube, at both ends, as shown in the figure, and makes with the axis of the tube an angle θ which is the "angle of contact." The weight of the drop may be neglected here, as its action is insignificant with regard to that of surface tension. When the pressure is equal at both ends of the drop, the meniscuses are identical, and so are the angles of contact θ and θ'. But if the pressure P at one end is increased, both meniscuses will alter their curvature. The meniscus in front of P will decrease its angle θ, which will tend toward zero, whereas the meniscus opposed to P' will increase its angle θ', which will tend toward 90°. The result of this deformation is to increase the force $T \cos \theta$ directed against P, and to decrease the force $T \cos \theta'$ directed with it. The difference will be a resistance against motion expressed by $T(\cos \theta - \cos \theta')$, which will draw closer to the limit T as the pressure is increased.

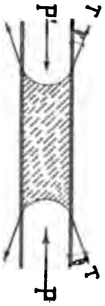

FIG. 7.

The same conditions would obtain in shales or clays capping a gas pool. There is nothing like a plane of separation between the gas of the pool and the water that fills the pores of the superincumbent rock; but there is a more or less irregular intermediate zone in which the gas and the water are commingled. The pores of the shale, by their juxtaposition, would constitute the Jamin tubes of the experiment, and these would be filled by the mixture, which would take the form of bubbles of gas intermingled with droplets of water.

Van Hise remarks[7] that "the majority of the particles of most clays, shales and slates are much smaller than 0.0012 mm., and therefore the openings of the rocks are subcapillary." He defines subcapillary openings as those which are 0.0002 mm. and less in diameter. Starting from these data, it is possible to calculate, with a sufficient degree of approximation, the thickness of the shales or of the clay which would seal up a given pressure. The writer has found that a few feet would be amply sufficient to seal up a pressure of 1,200 lb. per square inch.

It may be conceived that the same process would apply for the sealing of gas and oil pools laterally, along the dip and strike of the porous layers themselves, wherever the sealing is not already produced by changes in the nature of the "sand" from a pervious to an impervious one, or by the presence of water. Pore openings of sandstones are, for the most part, capillary; and should water not be present in the sand, oil may replace it for the Jamin tube effect. Calculation shows that the marginal zone thus constituted may reach a width of a few hundred feet. Pressure would decrease in this zone from the inside to the outside of the pool, progressively, a feature which is readily observed in the field.

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[7] A Treatise on Metamorphism, *Monograph*, 47, *U.S. Geological Survey*, p. 138 (1904).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Modern Methods of Mining and Ventilating Thick Pitching Beds

BY H. M. CRANKSHAW,* B. S., HAZLETON, PA.

(Arizona Meeting, September, 1916)

THE early methods of mining anthracite in the steep pitching Mammoth bed consisted in driving breasts up the pitch from the gangways and airways driven in the bed along the strike (Plate 2, Fig. 1). Breasts are simply rooms driven up the pitch the full thickness of the bed (Plate 2, Fig. 2), having a width of from 18 to 30 ft., and a manway or traveling way on one or both sides. (For detail of breast method see Whildin, *Trans.*, vol. 50, p. 704).

Many difficulties were encountered, and the maintenance cost of the timbered gangways was so great as to be practically prohibitive. To reduce this expense, the gangways were driven in the thin Skidmore bed underlying the Mammoth, and the coal mined through rock holes. (Plate 2, Fig. 2, *A*.)

It was found that after the breasts were driven up 80 to 100 ft., the coal almost invariably rushed or "ran away," making good ventilation almost impossible and the extraction of coal from the upper portion of the lift very difficult. The economic length of a lift is from 200 to 250 ft., and the percentage of extraction by the old method above mentioned was often under 50 per cent., while the coal recovered was so badly broken up as to reduce materially the percentage of large sizes.

Principles of Present Mining Methods

The present methods designed for the recovery of a maximum percentage of coal and a maximum yield of the prepared sizes depends largely on the following principles:

First.—Formerly the practice in mining bituminous coal was to space rooms so that as much coal as possible could be extracted in first mining (Plate 3). The small pillars were often lost, and an improvement in total extraction was found in the system of driving narrow rooms with wide pillars, with a much larger final recovery (Plate 4).

Second.—The early method of ventilation was to maintain a single current of air throughout the mine. This has been modified by systematically splitting the air and so establishing more efficient ventilation.

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* Manager, Harwood Coal Co.

Third.—The coal should be so mined as to allow the overlying measures to break and relieve the pressure, the break to be controlled and regulated by the method of mining so that it occurs where it is desired. This principle, which governs longwall mining in England, is fully recognized by the Consolidation Coal Co., and was well brought out in Mr. Grady's paper,[1] but in the type of mining under discussion it has rarely been applied. It has been claimed also by some mining engineers that the principle is inapplicable to the anthracite region, as the tough overlying strata would not break; that this conclusion is unwarranted is indicated by the fact that in longwall bituminous coal mining it is not unusual for several acres of roof to stand before the first break occurs, after which the roof breaks without further trouble.

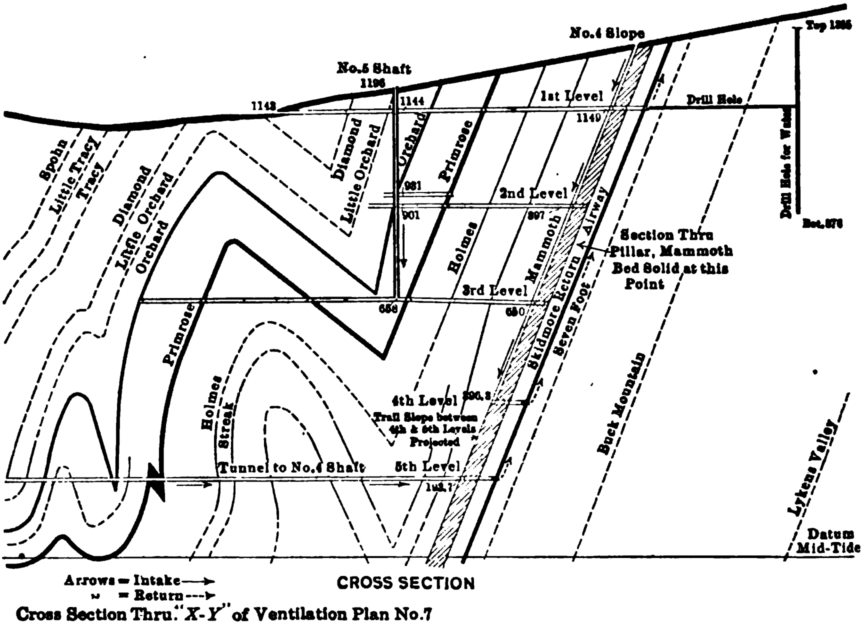

PLATE. 1.—METHOD OF MINING STEEP PITCHING MAMMOTH BED.
(Typical Section)

Therefore, we believe it has been proven that we must in first mining leave strong pillars; we must scientifically split the air to secure efficient ventilation; and we must so regulate the extraction of pillars that the roof breaks may be controlled.

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[1] *Trans.*, vol. 51, p. 138.

Disadvantages of Old Method

In the old method of mining the Mammoth bed, the coal was first partly worked over, after which the ground squeezed and settled through a period of years until the coal left behind had been so consolidated by pressure as to again allow of chutes being driven in the squeezed bed, and the original method of mining repeated, this process sometimes being gone over several times before the final extraction of the coal from the bed. The squeezing considerably shattered the coal and resulted in a much decreased percentage of the prepared sizes in the product of the second and subsequent minings.

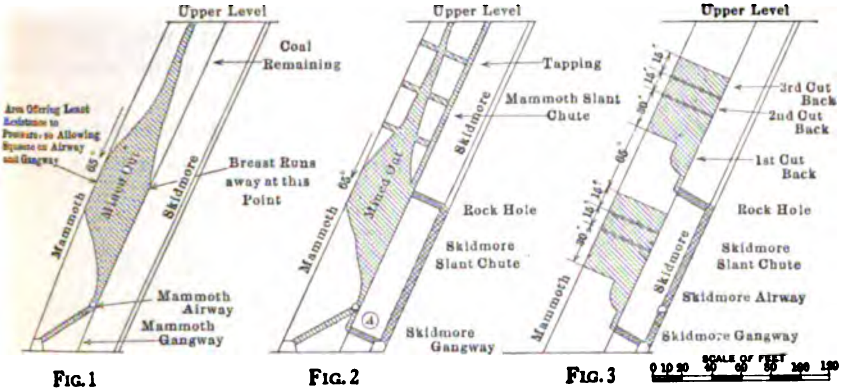

Fig. 1 Fig. 2 Fig. 3

PLATE 2.—OLD AND NEW METHODS OF MINING MAMMOTH BED.

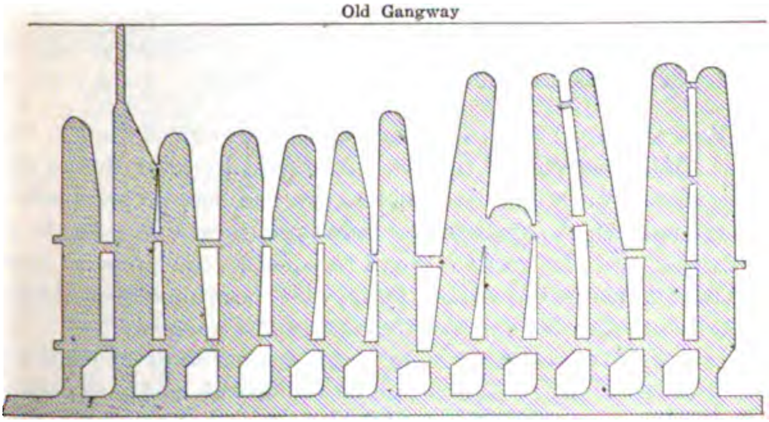

PLATE. 3.—SKETCH SHOWING PRIMITIVE METHOD OF MINING.

The modern methods aim to take out as much coal as possible in the first mining, and to avoid going over the same ground several times as was formerly necessary.

The method described is the result of 3 years of careful work and experiment, using at the same time all knowledge obtainable from the failures of the past (Plates 5 and 6). The paper by Mr. Whildin[2] described the methods developed for reworking mined-over ground. While doing this work constant studies were directed toward the development of a method which would be applicable to virgin ground, and which would overcome the defects of the old methods, the chief of which were: Low percentage of extraction; excessive cost of timbering; poor ventilation, and low percentage of prepared sizes, due to reworking the same ground after the coal had been crushed and squeezed.

It was found that after a breast had "run away" and been drawn until the rock from old workings appeared at the battery, the greater portion of the coal recovered had come from the lower part of the lift, Plate 2, Fig. 1. The upper part of the lift was practically solid, due to the fact that the space from which the coal had run had become practically filled with loose rock which supported the overlying measures. As the pillars were robbed and the ground·began to squeeze, the pressure naturally did the most havoc at the weakest point, which was found near the bottom of the breasts, or directly above the airway and gangway.

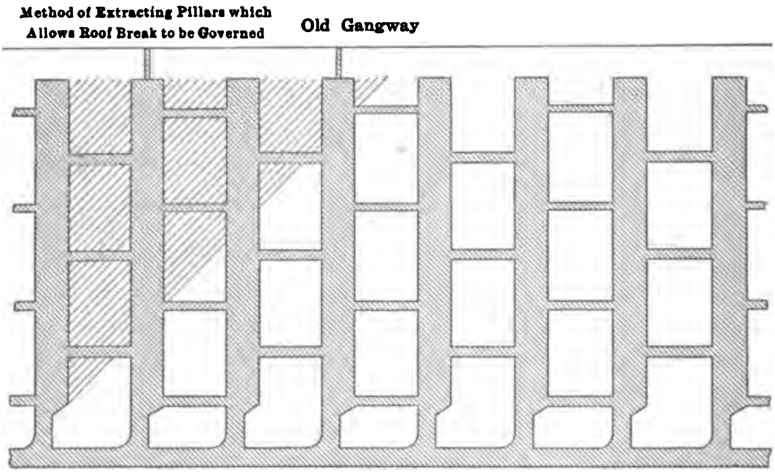
Method of Extracting Pillars which Allows Roof Break to be Governed
Old Gangway

PLATE. 4.—SKETCH SHOWING MODERN METHOD OF MINING. SCALE 1 IN. = 15 FT.

Since in most cases the mining was not done so as to allow a clean break of the roof, the pressure came on the airway and gangways in the form of a slow squeeze which broke the timber. This resulted in the conclusion that heavier timber was needed, and in many cases, sticks 24 to 30 in., and even 36 in. diameter were used for timbering purposes, resulting in an excessive timber cost.

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[2] *Trans.*, vol. 50, p. 698.

A careful study of Plate 2, Fig. 1, will show why the cost of timbering in Mammoth airways and gangways is excessive, and how the squeeze was brought on by the old method.

To reduce this timber cost the gangways were driven in the Skidmore bed underlying, instead of in the Mammoth (Plate 2, Fig. 2), involving the use of rock chutes from the Skidmore and the driving of slant chutes in the latter. The result was the development of a complete method of working the Mammoth bed from the underlying beds (Plate 2, Fig. 3).

Description of Present Method

In the working of a lift in the Mammoth bed 250 ft. long on a pitch of 65°, the bed being 40 ft. thick, the gangways and airways are first

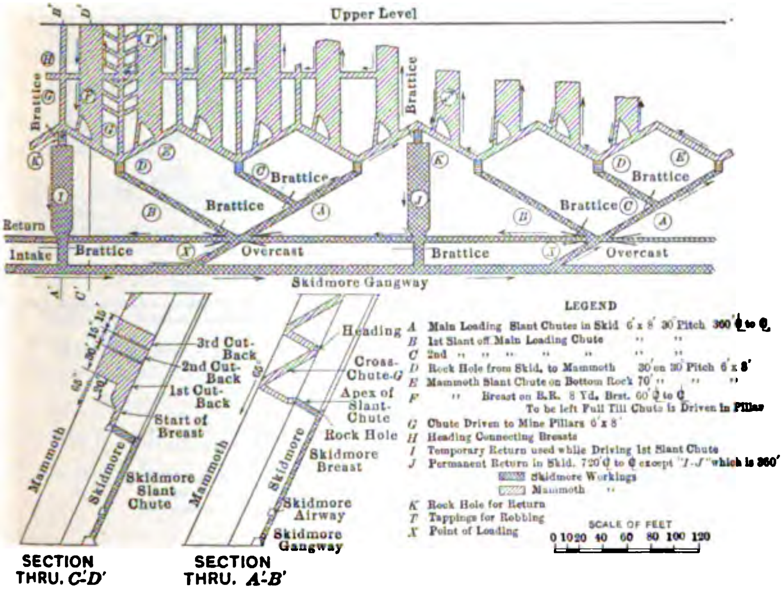

Plate. 5.—Present Method of Mining, Upper Section.

driven in the usual way in the underlying bed. Rock holes or chutes to the Mammoth from this gangway level are not driven until the workings in the underlying bed, following the plan to be described just below, have reached a point about halfway up the lift, Plate 5, with the result that the breasts in the Mammoth are restricted to a length of about 100 ft. The length of breasts was limited to 100 ft. because, as previously observed, past experience showed that the breasts almost invariably ran away when this length had been driven, making it practically impossible to carry to completion any system of working involving any greater length.

In the development of this plan, chutes *A*, *B*, *C*, Plate 5, are driven in the Skidmore bed, underlying the Mammoth, and rock holes *D* put through to the Mammoth 120 ft. apart, and from a point halfway up the lift. These rock holes are then connected by slant chutes *E* along the bottom rock of the Mammoth, from which the breasts are started on 60-ft. centers in the usual way. Each rock hole handles the coal from two breasts and the coal from three rock holes comes together at one loading chute at *X*.

It has been found desirable not to drive the rock chutes much in advance of the mining of the coal in the Mammoth, and to mine this coal as rapidly as possible. By this method after the work is once fully opened out the cost of the rock work is only about 20 per cent. of the total cost of timbering under the former method of mining.

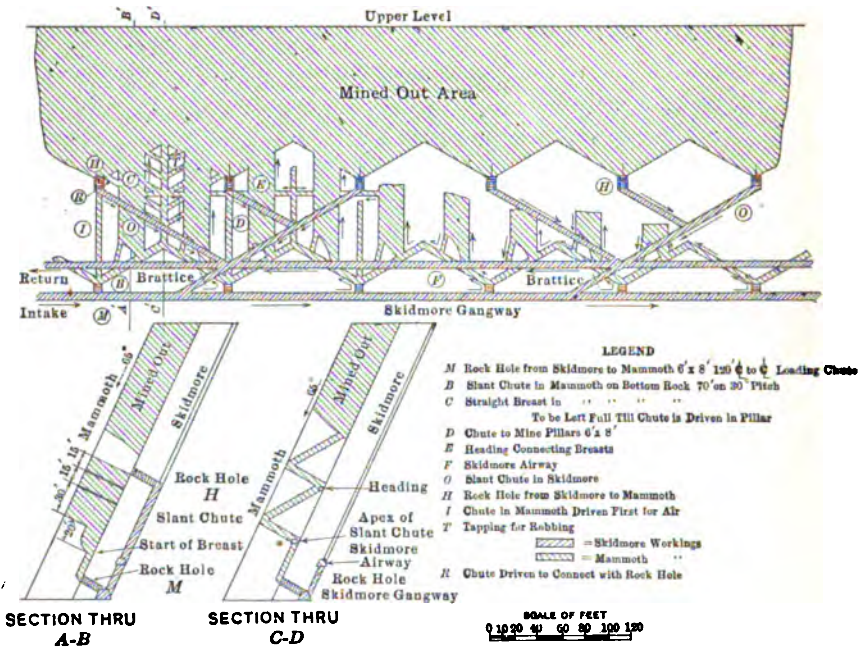

PLATE 6.—PRESENT METHOD OF MINING, LOWER SECTION.

Each set of six breasts has a separate split of fresh air and a ventilating system of its own. The ventilation is arranged very simply, on the split system, by driving a Skidmore breast *I* and *J* from the Skidmore airway far enough up the pitch so that a rock hole *K* put through from the top of the breast will connect with the apex of the slant chutes *E*. The main loading chutes *A*, 360 ft. apart, then become the main intakes, and as shown by the arrows, the air returns through rock holes *K* and Skidmore breasts *J* to the Skidmore return airway.

After the breasts are driven through to the upper level, cross chutes *G* are driven across the bed from the apex of the slant chutes *E*, for the purpose of robbing the pillars. The breasts are allowed to stand full until the cross chutes *G* are driven through to the upper level, and are drawn as the cross chutes *G* are worked down, the pillars being completely robbed by means of the tapping chutes *T*.

In this way all the chutes are driven in solid coal, the men are in strong ground all the time, thus being well protected, and a good supply of fresh air is always available to dilute any outburst of gas.

When the upper half of the lift has been robbed back, the lower half is worked in a similar way from the rock holes *M*, Plate 6, driven direct from the Skidmore gangway. These rock holes are connected by slant chutes *B* driven along the bottom rock of the Mammoth, and the whole operation repeated as for the upper half of the lift. Working the upper half first, however, has given an opportunity to ventilate the lower half easily, as follows:

As soon as the first breast *C* is driven up, a short chute *R* is driven from near the top of the breast to connect with the rock holes *H*; the slant chutes *O*, in the Skidmore bed, which previously were used as coal chutes, now serve as return airways, with no expense for upkeep.

The main chutes, wherever possible, should be driven 6 by 8 ft. so as to provide a good-sized manway for traveling and for carrying up timber. In order to secure a reserve pillar for fire protection and for limiting possible squeezing a large pillar is left every 720 ft., the length of each double section.

This method was first tried on the west side of the fifth level of No. 4 Shaft, Lansford Colliery, where the upper half of the lift in the first part of the work is now being robbed down. The results as to cost and extraction have been excellent.

Ventilation System

One result of the method of mining before described is to get all the main airways in strong permanent ground, so that they are not subjected to squeezing, and require a minimum expense for upkeep. Two mines producing 700,000 tons per year were working the section where the ventilation was remodeled, this work taking 2 years to complete. Two electrically driven Sirocco fans were installed, of 100,000 and 150,000 cu. ft. capacity, respectively, the larger one operated blowing and the smaller exhausting.

The mines were originally ventilated by fans located in the center of the property in the usual way (Plate 7). The air traveled along the main haulageways as intake airways and returned along return airways parallel to the main haulageways. As the mining concentrated at the extremities of the property, the following defects became apparent, resulting in unsatisfactory quality of air in the working places.

To get to the fans the air had to travel 2 miles; the main haulageways were so filled with traffic as the mines increased in production that their function as an intake airway was materially impaired; the return airways, where they were in the Mammoth bed, were exceedingly expensive to maintain, and where they were in the Skidmore, in many cases were too small to carry, for the required long distances, the volume of air needed.

By replacing the central system of ventilation with fans located near the extremities, the length of travel of the air was halved, and it was found practicable to abandon sections of the Mammoth airway which were expensive to maintain. In this way, an efficient method of ventilation was worked out, which made it possible to work coal which had been inaccessible for many years.

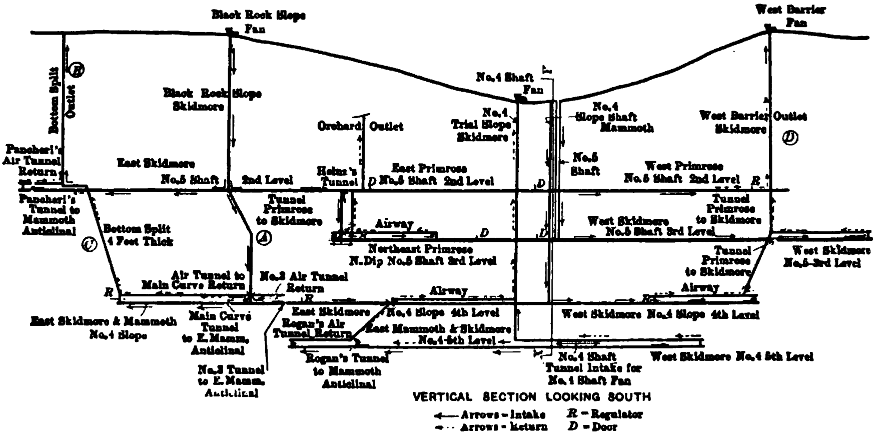
Black Rock Slope Fan
West Barrier Fan
Bottom Split
Outlet
Ⓑ
Black Rock Slope Skidmore
No.4 Shaft Fan
Orchard Outlet
No.4 Trial Slope Skidmore
No.4 Slope Shaft Mammoth
West Barrier Outlet Skidmore
Ⓓ
Pancheri's Air Tunnel Return
East Skidmore No.5 Shaft
2nd Level
Heinz's Tunnel
D
East Primrose No.5 Shaft 2nd Level
D
No.5 Shaft
West Primrose No.5 Shaft 2nd Level
R
Pancheri's Tunnel to Mammoth Anticlinal
Ⓒ
Bottom Split 4 Feet Thick
Tunnel Primrose to Skidmore
Ⓐ
Airway
D
D
West Skidmore No.5 Shaft 3rd Level
Tunnel Primrose to Skidmore
Northeast Primrose N.Dip No.5 Shaft 3rd Level
Tunnel Primrose to Skidmore
West Skidmore No.5-3rd Level
R
Air Tunnel to Main Curve Return
No.3 Air Tunnel Return
R
Airway
Airway
R
East Skidmore & Mammoth No.4 Slope
Main Curve Tunnel to E. Mamm. Anticlinal
East Skidmore
Rogan's Air Tunnel Return
No.4 Slope 4th Level
East Mammoth & Skidmore
No.4-5th Level
West Skidmore No.4 Slope 4th Level
No.3 Tunnel to E. Mamm. Anticlinal
Rogan's Tunnel to Mammoth Anticlinal
No.4 Shaft Tunnel Intake for No.4 Shaft Fan
West Skidmore No.4 5th Level
VERTICAL SECTION LOOKING SOUTH
←— Arrows - Intake
←·· Arrows - Return
R - Regulator
D - Door

PLATE. 7.—VENTILATING SYSTEM.

The diagram, Plate 7, shows the ventilating system employed in these mines, the central fan, by exhaust, ventilates the fifth or lower level, which is known as No. 4 Shaft level, and the "Black Rock Slope" force fan ventilates the second, third and fourth level on the east side, and the "West Barrier" exhaust fan ventilates the second, third and fourth levels on the west side.

By making the "Black Rock Slope" a blowing fan, it was possible to ventilate the section known as Pancheries' Tunnel, and the section known as the N. E. Primrose, without the necessity of having long return airways, the haulageways acting as intake airways for these sections.

When the three fans are in operation, the territory is so divided that there are a proportionate number of men to each fan, and the main air current is in each case divided into at least three main splits, each main split being again split up in the working places as required.

To date the adoption of the mining and ventilation methods described have given the following results:

In 1915, up to Oct. 30, 50,000 tons were mined by the new method, the labor cost being 19 c. per car lower than by the old method under similar conditions, and the saving in cost of timber at least 30 per cent.

While reliable figures as to extraction will not be available for about 2 years it is reasonable to believe that as the coal is "mined" instead of being allowed to run, less rock will be extracted, and the total car yield will be materially increased.

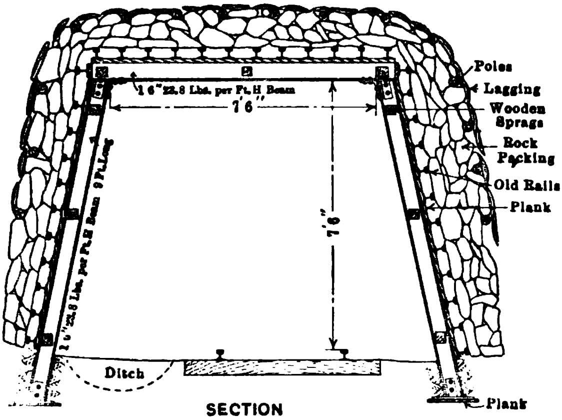

Plate. 8.—Method of Erecting Steel Timber in Mammoth Bed.

Use of Steel Timber

The use of steel timber is only commercially justified in gangways in the Mammoth bed when such gangways are not subject to squeeze due to mining operations in their immediate vicinity; but all gangways, turnouts and other places in the Mammoth bed, which are required for use over a period of many years, should be lined with steel timber properly erected.

For general use, it has been found advisable to use H-section steel for both legs and collars; 5- and 6-in. H-beams have been found most suitable for sets in gangways, with sizes for turnouts and special work as required by the conditions.

The method of erection is shown in Plate 8.

Figs. 9 and 10 illustrate typical examples of this method of timbering. In the gangways shown, the Mammoth bed is 60 ft. thick, very dry and very liable to run, the gangway being 1,000 ft. below the surface. The best of wooden timber rots out in 2 years.

Fig. 9 shows the gangway timbered up to the rock lying between the

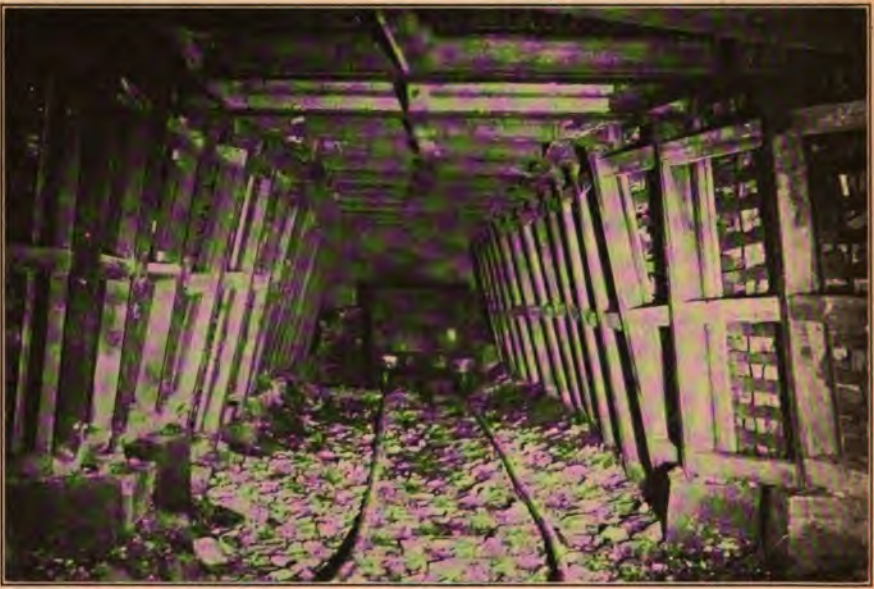

PLATE. 9.—GANGWAY IN WHICH STEEL "TIMBER" IS USED.

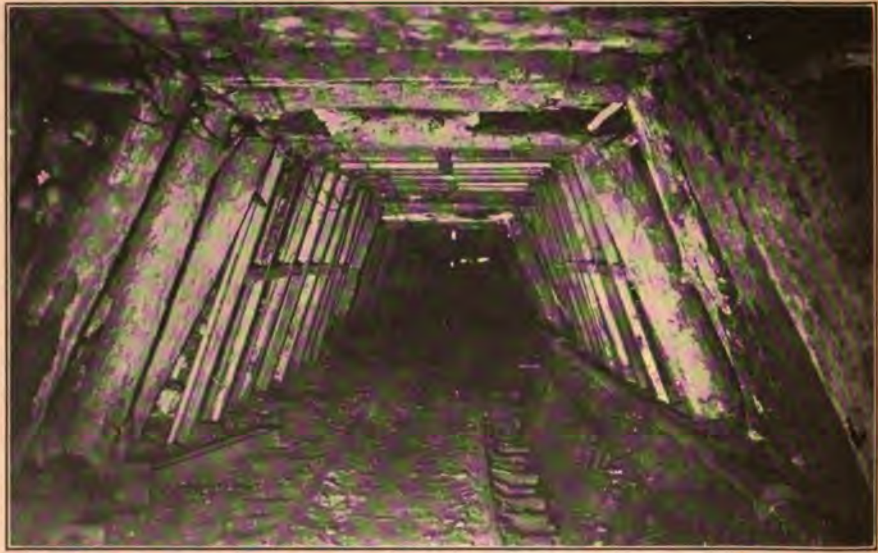

PLATE 10.—WOOD BEING REPLACED BY STEEL FOR GANGWAY SUPPORTS.

Mammoth and Skidmore beds. The tunnel was afterward driven without any of the timber being disturbed by the blasting of the rock.

Fig. 10 is an example of the replacement of wood by steel. Note the comparative size of wood and steel.

The author desires to express his appreciation of the assistance given by H. D. Kynor, John L. Richards and A. L. Walbridge in carrying out the work described and in preparing this paper.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Diesel Engines versus Steam Turbines for Mine Power Plants

BY HERBERT HAAS,* SAN FRANCISCO, CAL.

(Arizona Meeting, September, 1916)

CHEAP power is essential to large-scale mining and metallurgical operations, particularly where fine grinding of large tonnages has to be resorted to, as is the case with an increasing number of mines treating disseminated copper ores, or low-grade gold ores, in the grinding of which considerable power is consumed. A comparison of the cost of generating power with Diesel engines and steam turbines and a brief discussion of the different factors governing the profitable use of either type of prime mover may be timely in view of the importance of a cheap supply of power to the mining industry, and the prominence that has lately been given to Diesel engines in America.

As many of our most important mines and largest producers of metals are located in sections where neither cheap coal nor water power is available, and the price of oil fuel is increased by long hauls, the Diesel engine will become a prime mover of increasing importance. This applies particularly to the Southwestern States, where an ample supply of cold water for condensing purposes is unusual and local climatological conditions interfere in securing the high vacua essential to high turbine efficiency.

About 12 times as much water will be required for condensing purposes with steam turbines under general conditions as is needed for jacket-cooling of two-stroke cycle Diesel engine; four-stroke cycle Diesel engines require only one-twentieth of the cooling water.

Another condition favoring the use of Diesel engines is the high load factor at which most mine power plants operate. The influence of load factor on the cost of generating power is shown in Tables 1 and 2 and in Fig. 1.

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* Consulting Engineer.

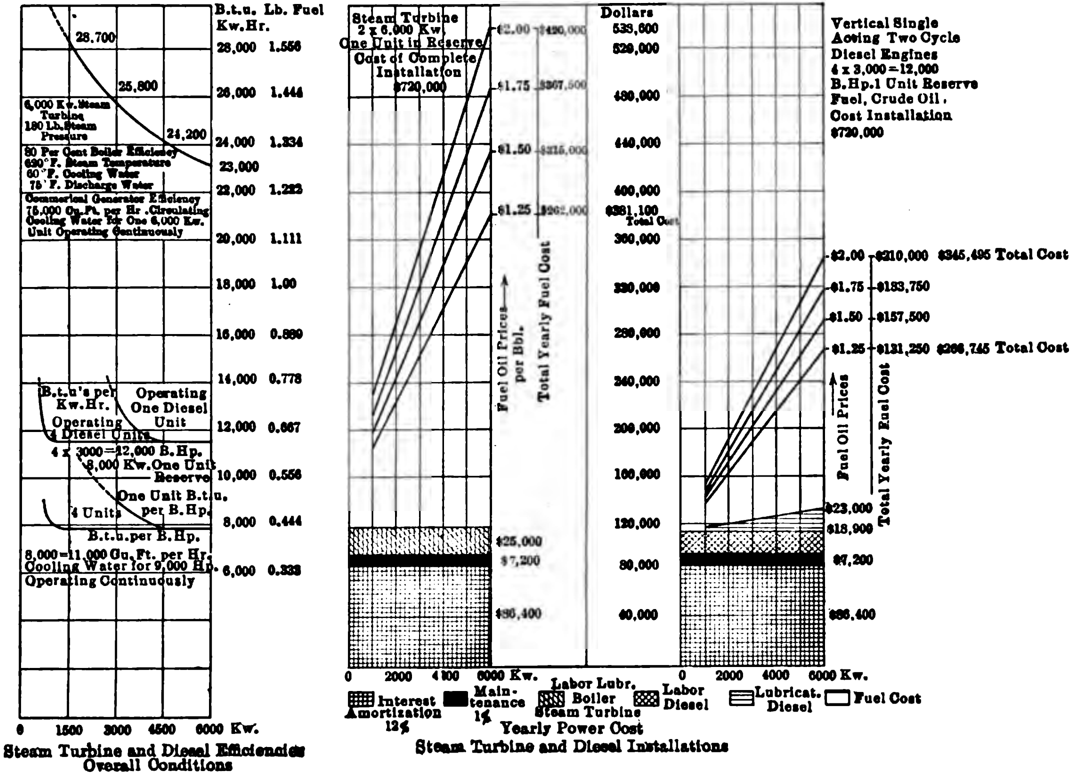

FIG. 1.—COMPARISON OF STEAM TURBINE AND DIESEL ENGINE EFFICIENCIES AND COSTS.

TABLE 1.—*Cost of Generating Power with Four* 3,000 *b.hp.* (2,000 *Kw.*) *Sulzer Diesel Engines Direct-Connected to Alternators and Exciters*

Costs are based on the following conditions:

Cost of complete plant			$720,000
Cost of fuel, $1.25 per bbl. of 320 lb.			
Cost of lubricating oil, 35 c. per gallon.			
		Per Month	
Labor— 1 chief engineer	at $200	$200	
3 watch engineers	at $150	450	
3 switch-board men	at $125	375	
3 helpers and oilers	at $ 90	270	
2 machinists	at $140	280	
12 men, total per month and year		$1,575	$18,900
Maintenance—1 per cent. of cost of installation, per year			7,200
Interest, at 6 per cent. per annum			43,200
Amortization at 6 per cent. and re-invested at 4½ per cent. to redeem capital of $720,000 in 15 years			43,200

The kilowatt-year, figured at 8,760 kw.-hr. Four, three, two, or one unit operating, respectively, furnish full, ¾, ½ or ¼ load, generating 70 million, 52½ million, 35 million and 17½ million kw.-hr. per year respectively. Three units to operate continuously, one unit to serve as stand-by. Cooling water in circulation for 9,000 hp. operating continuously, 8,000 to 11,000 cu. ft. per hour without re-cooling arrangements.

	12,000 hp. or 8,000 kw.		9,000 hp. or 6,000 kw.		6,000 hp. or 4,000 kw.		3,000 hp. or 2,000 kw.	
Kw.-hr. per year	70,000,000		52,500,000		35,000,000		17,500,000	
Barrels	140,000		105,000		70,000		35,000	
	Cost per Year		Cost per Year		Cost per Year		Cost per Year	
	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr. Cents
Fuel cost	$175,000	0.2500	$131,250	0.2500	$ 87,500	0.2500	$ 43,750	0.2500
Labor	18,900	0.0270	18,900	0.0360	18,900	0.0540	18,900	0.1080
Lubrication	30,660	0.0438	22,995	0.0438	15,330	0.0438	7,665	0.0438
Maintenance	7,200	0.0103	7,200	0.0137	7,200	0.0206	7,200	0.0412
Total direct	$231,760	0.3311	$180,345	0.3435	$128,930	0.3684	$ 77,515	0.4420
Interest and amortization	86,400	0.1234	86,400	0.1646	86,400	0.2468	86,400	0.4936
Total cost	$318,160	0.4545	$266,745	0.5081	$215,330	0.6152	$163,915	0.9356
Cost per kw.-yr	39.81	......	44.51	......	53.89	......	83.96	......
$1.50 oil	210,000	0.3000	157,500	0.3000	105,000	0.3000	52,500	0.3000
Total cost	343,160	0.5045	292,995	0.5581	232,830	0.6652	172,665	0.9856
$1.75 oil	245,000	0.3500	183,750	0.3500	122,500	0.3500	61,250	0.3500
Total cost	388,160	0.5545	319,245	0.6081	250,330	0.7152	181,415	1.0356
$2 oil	280,000	0.4000	210,000	0.4000	140,000	0.4000	70,000	0.4000
Total cost	$423,160	0.6045	$345,495	0.6581	$267,830	0.7652	$190,165	1.0856

TABLE 2.—*Cost of Generating Power with Two 6,000-Kw. Steam Turbines*

Costs are based on the following conditions:

Cost of complete plant	$720,000
One unit operates continuously at full load, one unit is stand-by.	
Cost of fuel oil, $1.25 per bbl. of 320 lb.	
Labor, lubrication, boiler upkeep, per year	25,000
Maintenance, 1 per cent. of cost of installation, per year	7,200
Interest and amortization, 12 per cent. of cost of installation, per year	86,400

Kilowatt-year figured at 8,760 kw.-hr. Turbines to operate at 180 lb. steam pressure, 620° F. steam temperature, 80 per cent. boiler efficiency, 60° F. inlet temperature, 75° F. discharge temperature of condenser circulating water; quantity of cooling water per 6,000-kw. unit : 74,000 cu. ft. per hr. Commercial generator efficiencies.

	8,000 kw.		6,000 kw.		4,000 kw.		2,000 kw.	
Kw.-hr. per year	70,000,000		52,500,000		35,000,000		17,500,000	
Barrels	300,000		210,000		150,000		83,700	
	Cost per Year		Cost per Year		Cost per Year		Cost per Year	
	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr., Cents	Per Kw.-hr.,	Per Kw.-hr., Cents
Fuel cost	$375,000	0.5357	$262,000	0.5000	$187,000	0.5357	$104,625	0.6000
Labor, etc	25,000	0.0357	25,000	0.0476	25,000	0.0715	25,000	0.1430
Maintenance	7,200	0.0103	7,200	0.0137	7,200	0.0206	7,200	0.0412
Total direct	$407,200	0.5817	$294,700	0.5613	$219,200	0.6278	$136,825	0.7842
Interest and amortization	86,400	0.1234	86,400	0.1646	86,400	0.2468	86,400	0.4936
Total cost	$493,600	0.7051	$381,100	0.7259	$305,600	0.8746	$223,225	1.2778
Cost per kw.-yr	61.75	......	63.50	......	76.60	......	111.70	......
$1.50 oil	450,000	0.6430	315,000	0.6000	225,000	0.6430	125,500	0.7171
Total cost	568,600	0.8123	433,600	0.8259	343,100	0.9818	244,200	1.3949
$1.75 oil	525,000	0.7500	367,500	0.7000	262,500	0.7500	146,375	0.8342
Total cost	643,600	0.9194	486,100	0.9259	380,600	1.0890	265,124	1.5120
$2 oil	600,000	0.8571	420,000	0.8000	300,000	0.8571	167,250	0.9513
Total cost	718,600	1.0257	538,600	1.0259	418,100	1.1962	286,050	1.6291

The last three double columns show the effect of increased cost of fuel oil on power cost, the total fuel cost being shown, as well as the total cost of power, which is merely increased in proportion to the increased fuel expenditure.

The higher the cost of fuel and the load factor, the greater will be the proportion of the fuel to the other costs making up the total cost of generating power. Capital charges (interest and redemption), maintenance, operating labor, and lubrication are practically constant amounts regardless of the load-output of the station; the fuel consumption and its cost are practically the only variables with the load.

The costs here given are based on conditions as generally found in the Southwestern States, and on Diesel engine operating experience in

European countries, where this type of prime mover has reached a high state of perfection, units upward of 4,000 hp. having proved entirely successful in operation.

This paper will not deal with questions of design, nor types of engines, as much space would be needed to do justice to the large number of excellent makes of engines. It is assumed that the reader has a general knowledge of Diesel engines.

The reasons favoring the selection of two 6,000-kw. steam turbines and four 2,000-kw. Diesel engine generating sets, can be briefly stated as follows:

1. Uninterrupted power service is essential, as failure of power would entail far greater losses in operating revenue than the interest on the cost of the reserve turbine unit.

2. The selection of two 6,000-kw. rather than three 3,000-kw. steam turbines is made on account of the greater efficiency of a 6,000-kw. over two 3,000-kw. units, the load to be supplied being nearly 6,000-kw. continuously. (In plants with fluctuating loads which can be supplied by one or more smaller units operating in parallel during different portions of the day, each unit operating at or near its full-load capacity, a number of small units is justified; such diversified load conditions, however, apply mainly to central stations in cities, rather than mine, mill, and smelter power plants, which have invariably a high load factor during the entire day.)

3. The reserve boiler capacity would be an additional third of the boiler capacity in continuous operation, regardless of size of turbines selected. The extra cost of the surplus turbine capacity by reason of installing two 6,000-kw. instead of three 3,000-kw. turbines is therefore moderate when taking future power costs and other factors into consideration.

4. The selection of four 2,000-kw. Diesel engine units is justified on the ground that larger engines are not any more efficient in fuel economy, and the relatively much higher cost of this prime mover[1] makes it policy to reduce reserve capacity to a safe minimum. An excessive number of small Diesel units should also be avoided, as the cost per kilowatt of installed power-plant increases, and the multiplication of cylinders, valves and engine parts to be taken care of increases maintenance and labor charges.

The heat utilization and cooling-water requirements of Diesel engines and steam turbines at full load are graphically shown in Fig. 2 for units of 2,000 hp.

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[1] A Diesel engine with its direct-connected compressor, alternator and exciter is a complete prime mover, and should be compared with a complete steam-power plant comprising boiler and condenser equipment, turbine and generator. Diesel engines are simple in that the installation is compact, comprising few parts, and occupying only about 50 to 60 per cent. of the space of a steam-turbine installation of like power.

In general, the selection of either type of prime mover will be governed by the following economic considerations:

1. Fuel is of chief influence on the total power cost where both the B.t.u. price[2] and load factor are high. These conditions favor the use of the Diesel engine.

2. Interest and redemption (amortization) are of chief influence on the total power cost with low B.t.u. price and low load factor; this applies particularly to stand-by plants, which are operated only occasionally or have to supply recurring peak loads. The installation cost of such plants must be kept as low as possible so as to avoid heavy capital charges distributable over a relatively small kilowatt-hours output of the station. Such conditions favor the steam turbine.

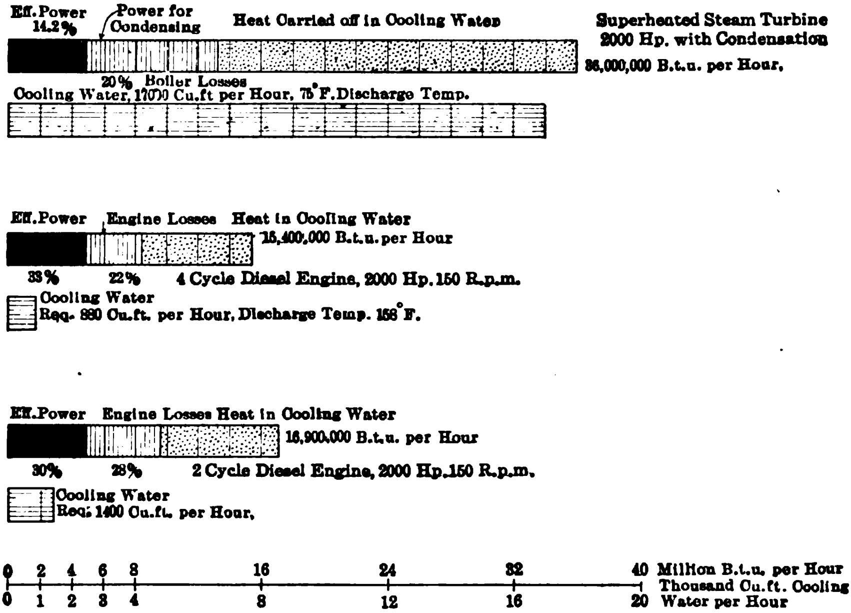

FIG. 2.—COMPARATIVE EFFICIENCIES OF STEAM TURBINES AND DIESEL ENGINES OF 2,000 EFFECTIVE HORSEPOWER.

3. Exceptions to (2) are cases where the constant and instant readiness of the Diesel engine give it preference, and installation cost is of secondary importance. Here we would have to balance the cost of keeping boilers under steam continuously against the difference of interest charges of steam-turbine and Diesel engine plants.

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[2] By B.t.u. price is meant the price or cost for a given number of B.t.u., say 1,000,000 B.t.u., all fuel prices being reduced to this standard of comparison.

4. Power plants located at the source of the fuel, either in the oil fields, or at the coal mine, and therefore enjoying the advantages of a very cheap fuel supply, will select prime movers costing least to install, *i.e.*, steam turbines, since fuel expenditures will weigh less than interest and redemption charges.

5. In many cases combination plants using Diesel engines for supplying the continuous and nearly constant main load, and steam turbines for furnishing periodically occurring peaks by the use of high-duty boilers with large water and steam spaces, capable of being forced when necessary, will prove most profitable. Thus, as an example, the periodical peaks produced in hoisting may be taken care of by a turbine floating on the line and operating in parallel with Diesel engines that supply the main load and operate constantly at or near full load.

6. Steam power will remain the cheapest power wherever waste-heat gases are available, as for instance gases from reverberatory smelting furnaces, where nearly one-half of the fuel used in smelting can be utilized for steam generation. Nearly 3,000,000 B.t.u. for every ton of charge smelted are thus available for steam generation, or about 150 hp.-hr. per ton of charge.

7. Up to capacities of 1,000-hp. steam turbines can compete with Diesel engines only in special cases, such as supplying exhaust steam for heating purposes. For such small units, particularly for greatly varying loads, high-grade reciprocating steam engines are preferable. For larger plants, from 1,000 to 10,000 kw. capacity, careful analysis must be made of the relative advantages of Diesel engines and turbines, a knowledge of the load factor, fuel prices and water conditions being necessary. Power plants larger than 10,000 kw. using units from 6,000 kw. upward, preferably use steam turbines, unless a combination of high load factor, high fuel cost, and poor water conditions favor a Diesel plant. This is a special condition frequently met.

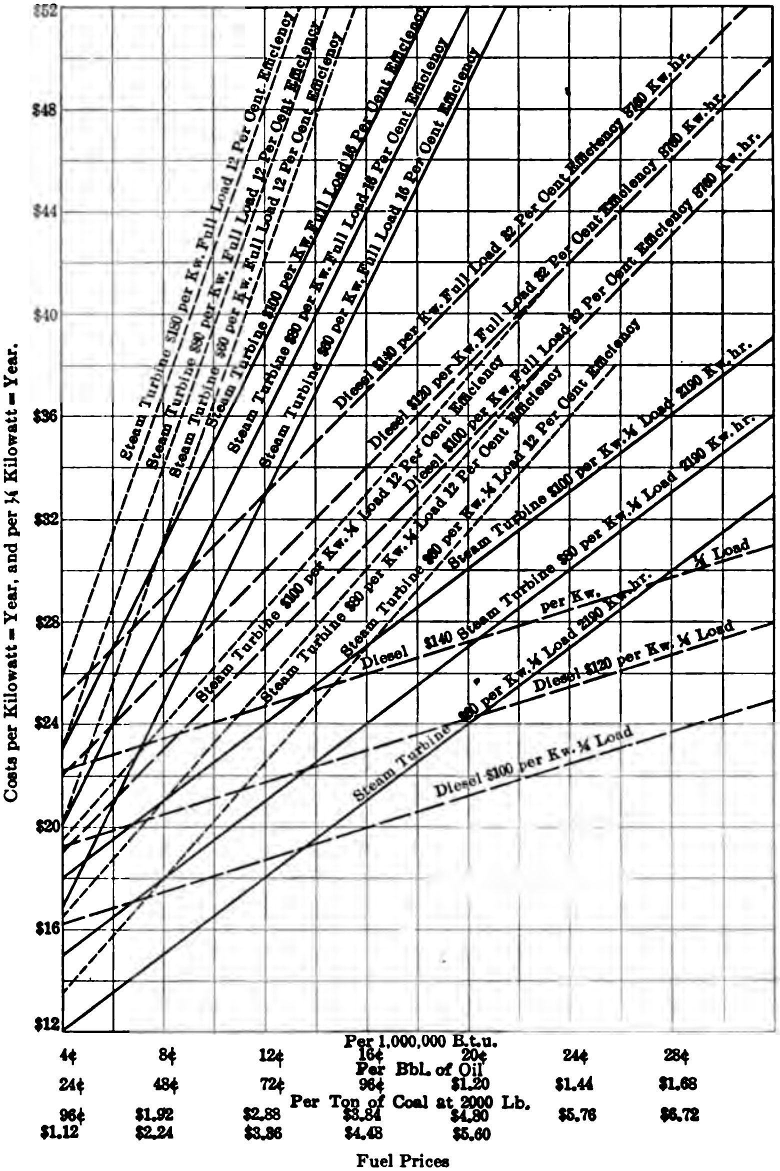
Fuel Costs

In comparing prices of different kinds of fuel, such as gas, coal, or liquid fuels, it is well to reduce all fuel prices to a common basis of absolute cost for 1,000,000 B.t.u. We have, then, for the cost of 1,000,000 B.t.u. transformed into mechanical work:

$$\frac{\text{Cost of 1,000,000 B.t.u.}}{\text{Over-all thermal efficiency of plant}}$$

and the fuel cost per brake horsepower-hour =

$$\frac{\text{Cost of 1,000,000 B.t.u.} \times 2{,}545}{\text{Over-all thermal efficiency} \times 1{,}000{,}000}.$$



Oil Costs Based on 1 Bbl. Containing 6,000,000 B. t. u.
Coal Costs Based on 1 lb. Containing 12,000 and 14,000 B. t. u.

Fig. 3.—Diagram Showing Comparative Costs of Steam Turbine and Diesel Engine Plants at Given Fuel Price and Load Factor.

Knowing the thermal efficiencies of different prime movers, we can then arrive at the absolute fuel cost of each per brake horsepower-hour with different kinds of fuel. Thermal efficiencies of steam engines and turbines vary from 6 per cent. operating simple and non-condensing to 16 per cent. with large units, depending on size, and operating conditions; and of Diesel engines from 30 to 35 per cent., at full load. Thus, for a steam plant with a 12 per cent. efficiency, an absolute heat price of 12c. for 1,000,000 B.t.u. comes to 100c. for every 1,000,000 B.t.u. turned into effective mechanical work, whereas for a Diesel engine plant with, say 30 per cent. efficiency, 1,000,000 B.t.u. transformed into effective mechanical work would cost only 40c., and the respective costs per brake horsepower-hour would be 0.2545c. and 0.1018c.

Where a supply of liquid fuel is available, the sound economic application of steam turbine and Diesel engine is determined by fuel prices, and load factor. The graph in Fig. 3 shows at a glance whether a steam turbine or a Diesel engine power plant is in place with a given fuel price and load factor. The abscissæ denote absolute fuel prices for each 1,000,000 B.t.u. These prices are translated into corresponding fuel prices per short ton (2,000 lb.) of bituminous coal having a fuel value of 12,000 and 14,000 B.t.u. per pound and per barrel of oil containing 6,000,000 B.t.u. The ordinates denote costs. The power costs represented by the curves include capital costs and fuel costs, but not operating labor, this being about the same for either type of plant, except for very large steam-turbine plants with units above 6,000 kw., which require less operating labor than Diesel engines and with which the latter can compete only under special conditions. The capital charges cover interest, amortization and maintenance, and are taken as 15 per cent. of the cost per kilowatt of installed plant capacity, which is for a complete, modern power plant with prime movers, switchboard equipment, and building, including necessary accessories, but without transformers or transmission line. The installation costs varying with size of plant and locality, three different costs per kilowatt of installed capacity are considered for each type of prime mover. For the turbine plant, $60, $80, $100, and for the Diesel plant $100, $120, and $140; 15 per cent. of these respective amounts are the capital charges added to the fuel costs, the curves marked "full load" showing the combined cost per kilowatt-year (8,760 kw.-hr.) of fuel and capital charges, the cost per kilowatt-year being shown in dollars in the ordinates.

The curves marked "quarter load" cover fuel cost for one-fourth kw.-year (or 2,190 kw.-hr) and the same capital charges as for full load; to arrive at the cost per kilowatt-year (8,760 kw.-hr.) *at quarter load*, the amount shown in dollars must be multiplied by four. This product divided by 8,760 will give the true cost per kilowatt-hour generated at quarter load.

Any other fractional load cost can be obtained by interpolation. An example will best illustrate the use of the graph: With a turbine plant costing $60 per kw. of capacity and a Diesel plant $100, the curve marked "steam turbine $60 kw. full load 16 per cent. efficiency," will be seen to intersect the curve "Diesel $100 per kw. full load 32 per cent. efficiency" at a fuel cost of 6c. per 1,000,000 B.t.u. and the cost per kilowatt-year (8,760 kw.-hr.) is $21. Under these conditions, the costs being equal for both prime movers, it would not pay to use a Diesel plant, a turbine plant having certain mechanical advantages. To the right of the intersection of these two curves, *i.e.*, with an increase in the price of fuel, the Diesel engine is the more economical prime mover. Assuming a 20c. fuel cost per 1,000,000 B.t.u., we find a cost of $35 per kw.-year at full load for the Diesel plant, and $49 for the turbine plant (16 per cent. efficiency), the difference of $14 representing the fuel saving per kilowatt-year of the Diesel plant under full-load conditions. A plant of 2,000 kw. would save $28,000 per year. The corresponding curves marked one-fourth load will be seen to intersect at a fuel cost of 13½c. per 1,000,000 B.t.u. and a cost of $19.20 for 2,190 kw.-hr. (¼ load). The kilowatt-year at quarter-load conditions would cost 4 × $19.20 = $76.80. In this case, Diesel engines would only pay if the fuel costs were higher than 13½c. per 1,000,000 B.t.u. The divergence to the right between these curves indicates the fuel saving above capital charges for the Diesel engine (or, to the left, saving in capital charges above fuel saving, with turbines) per kilowatt at quarter load. At 20c. for 1,000,000 B.t.u., this divergence shows ($24–$21.20) a net saving of $2.80 for the Diesel plant per kilowatt-year, for each 2,190 kw.-hr. per year (¼ load). For 75 and 50 per cent. load factor, the critical fuel prices can be ascertained by taking the difference between those at quarter and full loads, and adding two-thirds of this difference to the amount corresponding to quarter load to arrive at 75 per cent. load, and adding one-third of the difference to the amount for quarter load to arrive at 50 per cent. load. Any other critical fuel prices for other load conditions can be easily ascertained by interpolation.

It will be seen that curves based on 12 and 16 per cent. efficiency of steam-turbine plants are given. The higher figure is secured only with large units in sizes from 10,000 kw. upward. (These refer to the over-all efficiency of the entire power plant.) For smaller units, up to 2,000 kw., 12 per cent. represents good average practice. How a decrease in thermal efficiency favors the Diesel engine is strikingly shown by the curves for the steam turbine, with 12 per cent. efficiency at full and quarter load.

To secure complete power costs, the cost of operating labor and lubrication must be added to the above costs. The labor cost per kilowatt-year varies greatly with the size of the plant. Fig. 4 shows average labor operating charges per kilowatt-year at full load for different-sized Diesel engine plants. This labor charge remains practically constant, regardless of load variations, so that with fractional loads the labor increases in inverse proportion. In turbine and Diesel plants here considered, the labor cost is about the same for either type of plant, as any increased attendance required by Diesel engines is offset by firemen, water tenders, and usual operating force of turbine plants. The cost for lubrication is considerably higher for Diesel plants than for turbine plants, and amounts to from $2 to $4 per kilowatt-year, depending on the type of

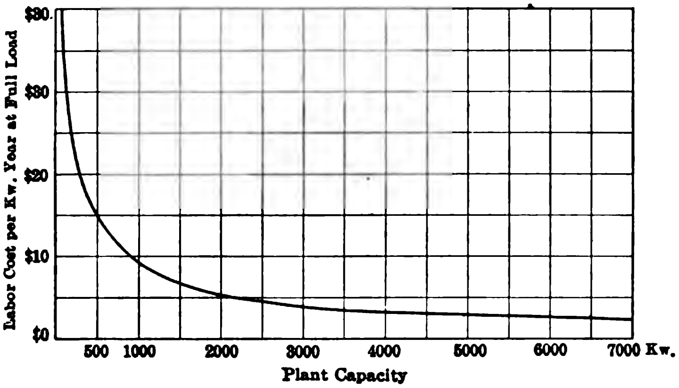


Fig. 4.—Operating Labor Cost Per Kilowatt-year (8,760 Kw-hr) At Full Load in Diesel Engine Plants.

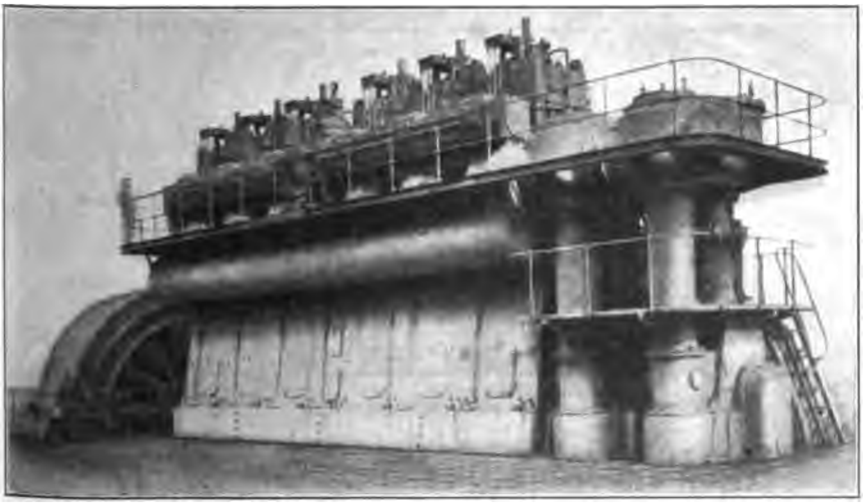


Fig. 5.—High-Duty, 4,000-Hp. Diesel Engine.

lubricating system used and the care of oilers. As a rule, a number of units are used in large Diesel installations, so that the lubrication expense is nearly proportional to the load, as with a decrease in load, units can be shut down. In turbine plants this greater expense for Diesel engines is partly balanced by the large amount of water required for condensing, which is 12 to 20 times greater than the supply needed for cooling Diesel engines. Only in plants with units in excess of 6,000 kw. have turbines decided advantages over Diesel engine plants.

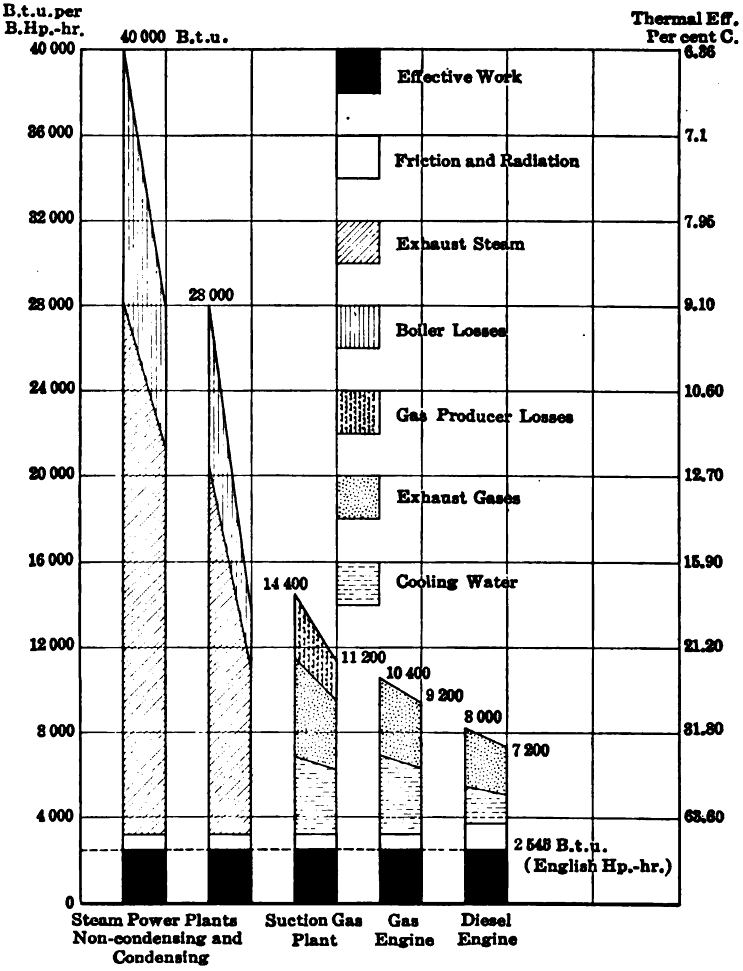


FIG. 6.—THERMAL BALANCE OF DIFFERENT HEAT ENGINES.

Fig. 5 is a good example of a high-duty 4,000-hp. Diesel engine. In conclusion, Fig. 6 shows the present status of different types of prime movers, expressed in thermal balances.

It is hoped that the above notes will be found helpful to those having to select the most economical prime mover to meet certain conditions, either turbines or Diesel engines having a wide field of usefulness; a combination of both types often best meets particular conditions. The selection should be governed by the above discussed economic considerations, the most important of which will always be the fuel price and load factor.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Comparisons between Electrolytic and Two Varieties of Arsenical Lake Copper with Respect to Strength and Ductility in Cold-Worked and Annealed Test Strips*

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NEW HAVEN, CONN.

(Arizona Meeting, September, 1916)

CHARACTER OF THE WORK IN HAND

In planning the present experiments, we have made a particular effort to secure that adjustment of working conditions which would render the forthcoming tests most serviceable by way of indicating the comparative properties of the three different commercial grades of copper used. More specifically, we have endeavored to obtain certain definite information with respect to the relative behavior under test of ordinary electrolytic copper and furnace-refined lake copper containing (*a*) a moderate percentage of arsenic and (*b*) a rather large percentage of arsenic, in addition to the usual content of silver and other impurities.

These comparisons were made systematic and comprehensive by testing throughout a series of (eight) reductions running from 0 to 70 per cent. in the case of cold-rolled metal, and throughout a series of (15) temperatures running from 100° C. to 1,000° C. in the case of annealed metal. In other words, a survey of properties similar to that made by Grard,[1] in the case of electrolytic copper, was here repeated (with modifications) and extended to two other commercial varieties of copper.

The great importance of copper both as a material of construction and as the base of our most important alloys (aside from steel) has led to innumerable investigations of its properties, particularly in commercial forms. Naturally, the merits of furnace-refined lake coppers *vs.* those of the very pure electrolytic product have furnished the subject of much experimental work and animated discussion, wherein the effect of arsenic has by no means been overlooked. In the literature of the past 10 years, we have found a number of papers in which the ordinary mechanical properties of various brands of copper have been compared after more or less limited or arbitrary forms of treatment, but no paper describing comparative tests after systematic variation of thermal and mechanical treatment. While much of this material is valuable, nevertheless, owing to its diverse and fragmentary character, we have not thought it desirable to use it as a general basis for discussion in connection with our own tests; or, indeed, to attempt any critical review of the literature bearing upon the present subject.

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[1] C. Grard: Cuivre et Laitons à Cartouches, *Revue d'Artillerie*, February–April, 1909.

It may be remarked that our experiments do not deal with the effect of arsenic in the absence of other common impurities—little or no progress appears to have been made in this direction since the publication of Bengough and Hill's experiments in 1910[2]—but refer exclusively to comparisons between electrolytic copper and two arsenical brands produced by the Michigan Smelting Co. throughout a wide range of treatment. It is our belief that the interests of consumers and other parties to whom comparative tests are of value may best be served by simple presentation of the results of these tests, leaving special questions of interpretation in the hands of any interested reader. Summaries of recent literature dealing with the properties and structure of copper in various forms may be found in Hofman's *Metallurgy of Copper*[3] and in reports of the Symposium on the Metallurgy of Copper, presented at the meeting of the International Engineering Congress, 1915, in San Francisco.[4]

It may also be appropriate to remark that we have restricted our work to the determination of tensile strength (maximum-load strength), elongation, and reduction of area—properties which are used as a basis for judgment as to quality affecting most engineering requirements. Since there are other criteria of judgment—some specific and capable of routine development, such as the determination of elastic limit, modulus of elasticity and of strength properties under different forms of loading, or at elevated temperatures; others not so well standardized, but, nevertheless, susceptible to quantitative treatment, such as electrochemical potential (rate of solution), varied chemical behavior, etc.; and still others which can only be gaged by manufacturing or service conditions, such as surface qualities, or peculiarities, during drawing operations—we have by no means exhausted the possibilities of comparison in the present series of tests.

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[2] G. D. Bengough and B. P. Hill: The Properties and Constitution of Copper-Arsenic Alloys, *Journal of the Institute of Metals*, vol. 3, pp. 34–97 (1910).

[3] McGraw-Hill Book Co., New York, 1914.

[4] Papers by William Campbell on the Metallography of Copper and by C. R. Haywood on the Physical Properties of Copper.

Questions of homogeneity with respect to the distribution of oxide and other micrographic features have been dealt with in a limited sense in the present paper, as have questions of resistance to the destructive agency of reducing gases during heat treatment ("gassing" of copper).

It has not been possible to investigate the effect of degree of deformation (reduction by rolling) and of time in coördination with temperature and strength properties in connection with the annealing experiments, since such an elaboration of the work would necessitate the performance of several thousand instead of several hundred individual tests. A general treatment of this subject has been attempted by one of us[5] and three-dimensional annealing diagrams (degree of deformation, percentage decrease in tensile strength, temperature for definite period of anneal; degree of deformation, percentage decrease in tensile strength, period of anneal at constant temperature), based upon a limited amount of experimental work in the case of "magnesium bronze" (copper, deoxidized with magnesium) have been given by von Müller.[6]

After due consideration of the questions raised above, we decided to use an annealing period of 40 min. and a degree of deformation amounting to a 50 per cent. reduction in area of section by cold-rolling as a basis for all annealing tests. Such procedure is not inconsistent with ordinary annealing practice and cannot fail to develop the principal annealing characteristics of the material in hand.

COMPOSITION OF THE METAL TESTED

Metal was sent to the rolling mill in the form of commercial cakes, 14 by 17 by 3 in., weighing in the neighborhood of 275 lb.

Two of these cakes were cast at the works of the Michigan Smelting Co., Houghton, Mich., and these represent the average quality of brands known as Mohawk and Copper Range copper, respectively. The other cake was obtained from The United States Metals Refining Co. ("D R W" copper) and is taken to represent electrolytic copper of average quality.

Determinations of oxygen and arsenic were made in the laboratory of the Michigan Smelting Co. Determinations of silver were made in the Hammond Laboratory.

Details of analytical procedure need not be introduced. All determinations were made by experienced chemists and we have every confidence in their accuracy. No determination was made of the very small quantities of iron, sulphur, nickel, cobalt, tellurium, etc., which may be present in the metal.

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[5] C. H. Mathewson: A Metallographic Examination of Some Ancient Bronzes. *American Journal of Science*, Series 4, vol. 40, pp. 550–555 (1915).

[6] von Müller: Beitrag zur Erkenntniss des Einflusses der Gluhdauer auf die Erweichung verschieded stark gereckter Leitungsbronze, *Metall und Erz*, Neue Folge, vol. 3, pp. 213–220 (1915).

The analytical results follow:

	Oxygen, Per Cent.	Arsenic, Per Cent.	Silver	
			Per Cent.	Ounces
Electrolytic copper	0.071	0.000	0.0005	0.15
Mohawk copper	0.052	0.096	0.069	20.0
Copper Range copper	0.055	0.296	0.052	15.0

Previous History of Test Pieces and Experimental Procedure

The experimental work was divided into two parts: (1) determination of the alteration in strength and ductility (strain hardening) produced by rolling the three kinds of material from an initially soft condition through a series of reductions as specified below, and (2) determination of the alteration in strength and ductility (removal of strain hardening) brought about by annealing the three kinds of material after a previous reduction of 50 per cent. in area of section by rolling.

Accordingly, two separate sets of test strips were prepared. In preparing both sets, the original cake was first planed to a thickness of 2 in., then hot-rolled and finally cold-rolled and annealed before variation in treatment designed to produce the two different sets of test strips was introduced. This procedure insured the production of test strips which had been thoroughly worked, both hot and cold, under conditions which are representative of the treatment commonly received by most mill products of this sort.

Preparation of the First Set of Test Strips

One of two methods may be selected in preparing a set of test strips of different tempers (set 1 according to the above designation), *i.e.*, the strips may be finished at a succession of gages, or all of them may be finished at the same gage.

Steps (*a*), (*b*), and (*c*), of the accompanying diagram, Fig. 1, represent the preliminary stages of rolling which may be considered common to both methods. The relative changes in thickness from one stage of reduction to another are correctly shown in the sketches and the actual gages and percentage reductions between stages are written into the sketches. Sketch (*a*) represents the original cake planed to a thickness of 2 in.; sketch (*b*) represents the material after hot-rolling to gage No. 0, B. & S. (0.325 in.), a reduction of 83.75 per cent.; and sketch (*c*) represents this material after cold-rolling to gage No. 8 (0.128 in.), a reduction of 60.6 per cent. At this point the metal is annealed.

If it is desired that the final product shall be obtained according to the first method, the annealed metal (*c*) is cut into strips which are rolled to a series of measured reductions, in which case the thickness will decrease progressively from that of the first strip (minimum reduction) to that of the last (maximum reduction).

This method was adopted in the present investigation and the numerical specifications sent to the mill[7] to cover this part of the work are written into sketch (*d*) of Fig. 1, which represents the series of eight strips composing this set. For convenience, even B. & S. gage numbers running from 8 to 18 (exception, 8½) were used to limit the separate stages. The percentage of the initial thickness (0.128 in. in all cases) taken off in

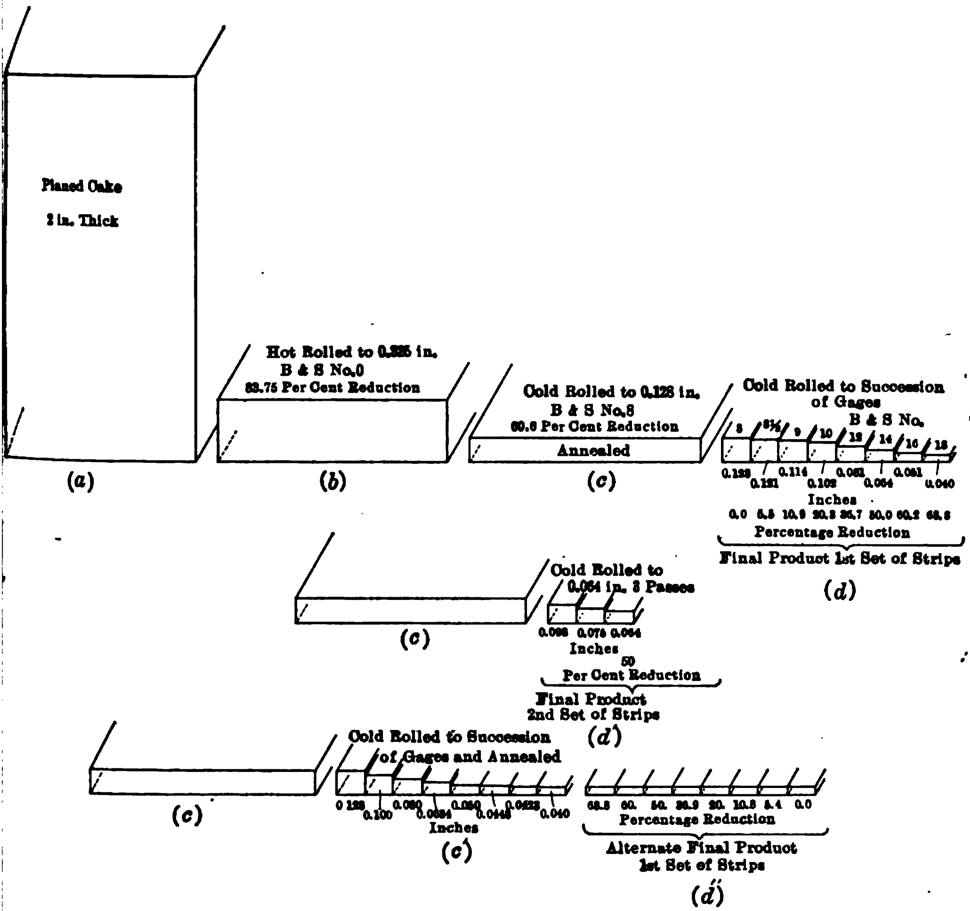

Fig. 1.—Diagram to Represent Preparation of Test Strips.

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[7] The material used in all of the tests described in this paper was rolled by the Bridgeport Brass Co.

the reduction is entered below the value for the final thickness under the sketch of each respective strip. Thus, in the case of the last strip:

$$\frac{0.128 \text{ in. (initial thickness)} - 0.040 \text{ (final thickness)}}{0.128 \text{ in. (initial thickness)}} \times 100 = 68.8.$$

In view of the convenient survey of the rolling stages afforded by the sketches of Fig. 1, it appears superfluous to draw up a table of the data involved.

If it is desired that the final product shall be obtained according to the second method (which was not used in these experiments), stages (*c′*) and (*d″*) may be substituted for stage (*d*). According to (*c′*), the metal from (*c*) is cut into strips which are given a series of reductions as shown, so that, after annealing the lot, another set of reductions will produce test pieces of the same thickness, but different tempers, as indicated in (*d″*). In order to make the comparison perfectly clear, the reductions marked in sketch (*c′*) have been calculated so as to yield the same series of reductions after the strips have been rolled to a final product of uniform thickness (0.040 in.) as was obtained in the final product according to the first method. These reductions are marked in sketch (*d″*).

Since we had to consider carefully the respective advantages of these two methods, it seems appropriate to devote some space to a consideration of this subject. It is unquestionably desirable, other things being equal, to perform all tests upon strips of the same cross-sectional area. The relation between surface and volume of the test piece influences to some extent the testing value calculated per unit of sectional area. There appears to be a "skin effect" which tends to produce higher values of tensile strength in thin specimens than in thick specimens. Notably, in very thin specimens, the testing results are considerably dependent upon the surface conditions of the test piece and comparatively insignificant scratches, or indentations, may lead to premature rupture.[8]

There are, however, certain difficulties in the way of producing a satisfactory set of test strips of uniform thickness and varying percentages of reduction. In the first place, the different samples must receive different amounts of work before the necessary anneal which precedes the final impression of a graded series of reductions bringing all strips to the same finishing thickness. This means that the micrographic characteristics, notably grain size, of the different strips cannot be uniform; those which received a heavy reduction before the anneal will possess a more highly refined grain than those which received a light reduction. It is true that we may depart somewhat from the order of changes sketched in (*c*) and (*c*′) of Fig. 1 by cold-rolling in (*c*) to a lesser degree, say, to gage 0.220 in., instead of gage 0.128 in., omitting the anneal at this point, and continuing the rolling so as to produce strips of the gages shown in (*c*′). All of these strips would then have received comparatively heavy reductions and grain refining would be more uniform in the ensuing anneal. This procedure would modify but not entirely remove the difficulty in question.

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[8] Metallographists are generally familiar with the phenomenon of surface flow, first demonstrated by Beilby, which produces a surface hardening, quite marked in the case of soft metals such as copper, during any ordinary process of surface dressing or finishing.

From a practical standpoint, there would be no check on the rolling-mill work, since the different gages in the intermediate product are all reduced to the same gage in the final product and any departure from specifications in the intermediate work could not be recognized in the final product. This is serious, because such inaccuracies might greatly influence the temper of the product, particularly in the case of light reductions. It may be noted, in this connection, that to produce a final product of gage 0.040 in., having a reduction of 5.4 per cent., the strip is required to be only 0.0423 in. thick before rolling. This requirement is too rigid for ordinary mill work. When the same reduction is made according to the first method, it is performed upon a thicker strip and 0.007 in., instead of 0.0023 in., is taken off. To eliminate this difficulty, the whole set of strips would have to be finished thicker.

In the case of the first method, we are in possession of a reliable check upon the rolling-mill work, since the amount of reduction shows directly in the individual strips and may be measured up with the micrometer.

On the basis of these remarks, it is perfectly clear that the first method is preferable when the metal must be sent to a rolling mill for preparation. With respect to the shortcomings of metal prepared in this way, it may be stated that from the standpoint of comparison between different brands of metal, as in the present case, such very largely disappear, since they only affect the comparison between different tempers and the validity of the succession of testing values thereby secured. It is here assumed that the different kinds of metal follow the same schedule in the mill.

Preparation of the Second Set of Test Strips

The test strips of the second set were finished at gage No. 14 (0.064 in.) with a reduction of 50 per cent. taken off in three passes beginning with annealed metal at gage No. 8 (0.128 in.). These strips were taken from the same cakes as the first set of strips. The rolling operations are, accordingly, those shown in Fig. 1, (*a*), (*b*), and (*c*) followed by (*d*′), instead of (*d*).

Specifications

It has already been stated that the numerical specifications for rolling are indicated in the sketches of Fig. 1. In sending these specifications to the mill, it was requested that the rolling conform as closely as possible to the data submitted, but a margin of 0.003 in. above or below any given gage was allowed. The mill work was conducted under the supervision of the works laboratory and we were generally satisfied with the product.

Slight departure from the specifications scarcely affects the percentage reductions figured from the specifications except in the case of very light reductions, as in the first two of the first set. Here, the actual reductions varied considerably in the three kinds of material, which would not have been the case if each had been passed through the same setting of the rolls. This was recommended but did not appear feasible in the mill. However, such variations are not serious, since we have the data for calculating the exact reductions obtained. All of the data in question will be found in Tables 2 and 3.

Since we are chiefly interested in comparisons between the three kinds of material, it is important that all receive the same treatment in prescribed annealing operations. On this account, it was insistently urged that the different lots be annealed simultaneously and in the same part of the load. From our micrographic examination, we believe that these specifications, which called for uniform anneal for 1 hr., at 650° to 700° C., were properly carried out.

Procedure with the First Set

In the case of the first set of test strips, the laboratory work consisted solely in testing the prepared strips.

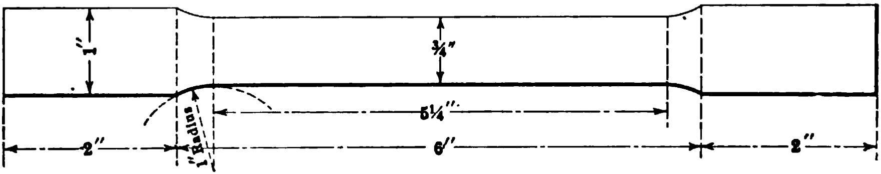


FIG. 2.—DIMENSIONS OF TEST STRIP.

For this purpose, a 50,000-lb. Riehle Universal testing machine was used. Each final result represents the average of tests on four separate strips. A uniform rate of loading was used in all cases, light scratch marks instead of punch marks were made on the strips, and it is believed that all essential precautions were observed in the interest of obtaining the most trustworthy results. Test pieces were accurately milled according to the accompanying sketch (Fig. 2). It may be noted from this sketch that the original strips were 10 in. long by 1 in. wide, while, in milling, three-quarters of the original sectional area was retained in the test section. Our intention in this respect was to cut away just enough material from the sides to insure a properly located break without decreasing the breaking load to a point which would render the errors at the machine serious in comparison. The thinnest (most severely reduced) strips (0.040 in.), although the strongest per unit of sectional area, broke under the lightest load, viz., some 1,700 lb. The minimum breaking load for the regular annealed strips of the second set (0.064 in.) was some 1,250 lb., while the weakest of the small number of strips tested after anneal in a reducing atmosphere broke at 930 lb. Thus, if the machine can be read accurately to 10 lb., the maximum testing error is only about 1 per cent.

Breaks were considered faulty when they could not be placed near the middle of a 2-in. length selected with the aid of the scratch marks along the milled section of the strip, viz., elongation was measured in 2 in. and the measurements were made with proper regard for the distribution of elongation along the broken strip.

Measurements of reduction of area were made with the micrometer and, owing to the comparatively small sectional area of the strips, the results cannot be considered very accurate, particularly in the case of all values below about 25 per cent.

Procedure with the Second Set

The strips of the second set were all tested after a prescribed annealing treatment. The following is a summary of the methods adopted in this part of the work: The annealing was done in a ribbon-wound alundum tube 18 in. long and 1½ in. in inside diameter. This resistor was covered with alundum cement and mounted in a rectangular box made of transite asbestos wood, as shown in Fig. 3.

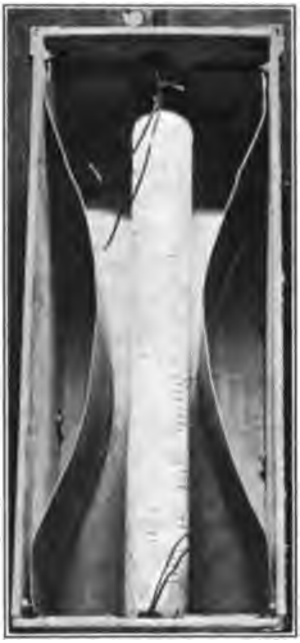
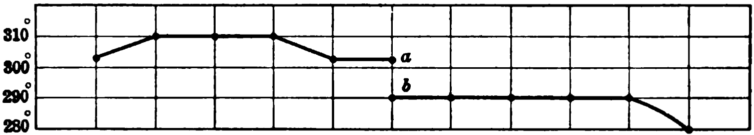


FIG. 3.—INTERIOR OF ANNEALING FURNACE WITH FILLING REMOVED.

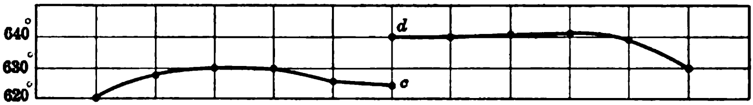
In order to flatten the thermal gradient in the tube, the heat capacity of the furnace was increased by means of a loose filling of granular fire clay, and false inner sides bent as indicated in the figure were used to cut down the quantity of filling around the central portion of the tube (which would ordinarily run hottest). In this way, the loss of heat along the sides of the furnace was made to decrease toward the end-regions, thereby compensating for the extra loss at the ends of the furnace.

The gradients along the central 10 in. of the tube, in which the test strips were located, are shown in Fig. 4 at three different temperatures in different parts of the annealing range. It may be noted that the center of the furnace was kept at a constant temperature (the furnace was allowed to come to equilibrium with a fixed energy input), while the gradients were taken on one side, and again, at a constant temperature in another series of tests (not exactly the same temperature as before), while the gradients were taken on the other side. Interpreted in this sense, these curves show a maximum variation from center to ends (central region of 10 in.) of 15° at approximately 300°; 11° at approximately 600°, and 22° at approximately 900° C. These gradients are made still flatter by the balancing effect of the load.

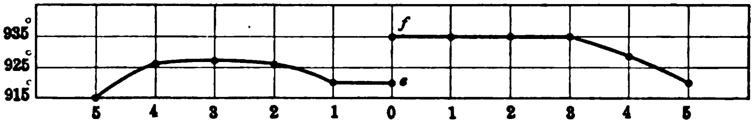
Four milled strips were annealed at once. It was therefore necessary to make three runs at a given temperature in order to handle the 12 strips



Curve(*a*) Center of Furnace Constant at 305° C. Curve(*b*) Center of Furnace Constant at 290° C.



Curve(*c*) Center of Furnace Constant at 625° C. Curve(*d*) Center of Furnace Constant at 640° C.



Curve(*e*) Center of Furnace Constant at 920° C. Curve(*f*) Center of Furnace Constant at 935° C.

FIG. 4.—TEMPERATURE GRADIENTS OF ANNEALING FURNACE AT APPROXIMATELY 300°, 600° AND 900° C.

required (four each from three kinds of metal). We were able to make these different runs under practically identical conditions, but, in anticipation of possible variations of treatment, each load was composed of strips from the different kinds of metal, so that any differences shown by the average of tests on the several kinds of metal could not be ascribed to differences in annealing treatment. Marking the electrolytic samples 1, the Mohawk samples 2, and the Copper Range samples 3, the different loads were made up as follows:

First Load	Second Load	Third Load
1—2—3—1	2—3—1—2	3—1—2—3

In starting an anneal, the furnace was brought to a temperature some 10° above the actual annealing temperature sought and then the four strips were pushed into the center of the furnace tube side by side with the end of the thermocouple tube as nearly as possible in the center of the load. When the temperature reached the desired value, attention was paid to the resistances in series with the furnace tube and such alterations as were necessary to keep the temperature constant were made from time to time on the basis of 5- or 10-min. observations of temperature.

The galvanometric method of measuring temperature was employed in connection with an ordinary platinum, platinum-rhodium thermocouple (Siemens and Halske galvanometer of 410 ohms internal resistance). The hot junction of the thermocouple remained in perfect condition throughout all of the experiments. Standardization was made at the freezing points of pure tin, 232°; zinc, 419°; antimony, 631°; and silver (in hydrogen), 961°, keeping the cold junction at 20°. The same conditions were observed in making all anneals. An empirical correction curve of the temperature scale of the galvanometer was made on the basis of the standardization results and all temperatures were corrected by interpolation. Re-standardization of the apparatus after completion of the experiments showed that constancy had been maintained within 10°.

A typical annealing curve shows a preheating period of about 20 min., during which the temperature rises gradually to the value chosen for annealing, and an annealing period of 40 min., during which the temperature varies only a few degrees. Owing to the uniformity which was attained in making these anneals, it does not appear desirable to devote space to an elaborate tabulation of the annealing data. We believe that the annealing temperatures reported are consistently accurate (making allowance for all sources of error) within 15°.

TABLE 1

Temperature of Anneal, Degrees Centigrade		Preheating Period in Minutes			
		Load No. 1	Load No. 2	Load No. 3	Average
100 water		4	2	2	2.7
200 oil		4	4	3	3.7
250	Electric furnace	21	22	20	21.0
275		20	20	21	20.3
300		19	16	17	17.3
325		22	22	23	22.3
350		19	19	19	19.0
400		21	18	20	19.7
500		19	15	14	16.0
600		18	18	19	18.3
700		17	18	18	17.7
800		15	15	15	15.0
900		12	12	13	12.3
1,000		13	12	18	14.3

With respect to the preheating periods, which were naturally subject to more or less variation in the different experiments, a summary of the principal data appears to be desirable and this is given in Table 1.

Some of the strips of the second set were annealed in an atmosphere of illuminating (coal) gas. The approximate composition of this gas by volume is as follows:

	Per Cent.
Hydrocarbons	48.5
Hydrogen	25.0
Carbon monoxide	18.0
Carbon dioxide	2.0
Oxygen	0.5
Nitrogen	6.0

These anneals were made in a furnace similar to the one previously described, in which an iron tube was substituted for the alundum tube and both ends were closed except for a small gas inlet at one end and exit at the other end. The thermocouple was introduced at intervals through a suitable hole at one end and left in the furnace long enough to reach the temperature of its surroundings, after which it was withdrawn to avoid contamination by long exposure to the reducing gas, and the hole plugged up. Less accurate control was secured in these experiments and the preheating periods were not determined.

Strips from the different kinds of metal were mixed in each load and, aside from all questions as to the ultimate accuracy of the temperature measurements, valid comparisons between the different kinds of metal (which was the main object of all tests) were obtained.

Samples for micrographic examination (1 by 1 in.) were cut from the ends of the test pieces after annealing.

TABULATION OF RESULTS

(1) *Tests in the Cold-Rolled Condition. First Set of Strips*

Table 2 shows the results of tests made on strips rolled to a succession of gages beginning with No. 8 (B. & S.), viz., 0.128 in., and ending with No. 18, viz., 0.040 in. Test strips were milled according to Fig. 2. Micrometer measurements are not given in the table except in case of the thickness. The latter is of importance in indicating character of the the product with respect to temper and an average must be used in calculating the percentage reduction of area effected by rolling, while other measurements are of a routine character, as prerequisites to the calculations of strength and ductility.

The previous history of the strips is given in the sketches of Fig. 1.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips*

Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.

No.	Brand of Copper and Reduction by Rolling*	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
1	Electrolytic copper, reduced 0.0 per cent. by rolling.	0.1315	33,100	40.5	52.0
2		0.1310	32,900	41.0	52.0
3		0.1310	32,500	41.9	50.0
4		0.1310	32,900	40.6	51.5
	Average...........	0.1310	32,850	41.0	51.4
5	Mohawk copper reduced 0.0 per cent. by rolling.	0.1280	32,700	43.7	51.5
6		0.1285	33,700	44.0	54.5
7		0.1280	33,500	45.5	54.5
8		0.1290	33,100	45.0	54.0
	Average...........	0.1284	33,250	44.6	53.6
9	Copper Range copper reduced 0.0 per cent. by rolling.	0.1320	33,100	43.5	53.5
10		0.1310	32,400	43.1	53.0
11		0.1315	33,100	45.0	53.0
12		0.1320	33,100	45.2	53.0
	Average...........	0.1316	32,925	44.2	53.1
13	Electrolytic copper reduced 6.9 per cent. by rolling.	0.1220	34,800	39.3	46.0
14		0.1220	34,300	38.0	45.0
15		0.1220	34,800	38.7	42.0
16		0.1220	34,700	40.0	44.5
	Average...........	0.1220	34,650	39.0	44.4
17	Mohawk copper reduced 6.2 per cent. by rolling.	0.1200	34,200	44.3	45.0
18		0.1200	34,600	44.0	44.0
19		0.1205	35,200	44.1	42.5
20		0.1210	34,100	44.5	45.0
	Average...........	0.1204	34,525	44.2	44.1
21	Copper Range copper reduced 8.7 per cent. by rolling.	0.1200	35,500	39.5	38.0
22		0.1205	35,600	36.2	34.0
23		0.1200	35,100	40.6	38.0
24		0.1200	35,600	39.3	38.0
	Average...........	0.1201	35,450	38.9	37.0
25	Electrolytic copper reduced 12.9 per cent. by rolling.	0.1140	37,300	35.9	34.0
26		0.1140	36,900	36.2	35.5
27		0.1140	36,800	35.8	34.0
28		0.1145	36,100	35.8	34.0
	Average...........	0.1141	36,775	35.9	34.4
29	Mohawk copper reduced 11.1 per cent. by rolling.	0.1140	36,800	40.2	36.0
30		0.1145	36,600	40.7	36.5
31		0.1140	37,000	40.0	38.0
32		0.1145	37,100	39.8	38.0
	Average...........	0.1142	36,875	40.2	37.1
33	Copper Range copper reduced 13.1 per cent. by rolling.	0.1145	38,100	33.7	31.0
34		0.1145	38,300	33.7	29.0
35		0.1140	36,900	32.2	27.5
36		0.1145	37,500	32.3	31.0
	Average...........	0.1144	37,700	33.0	29.6
37	Electrolytic copper reduced 21.4 per cent. by rolling.	0.1030	40,900	23.4	13.5
38		0.1030	41,400	22.5	12.5
39		0.1030	41,000	22.5	9.5
40		0.1030	41,000	24.4	16.5
	Average...........	0.1030	41,075	23.2	13.0
41	Mohawk copper reduced 20.2 per cent. by rolling.	0.1025	41,600	27.3	16.0
42		0.1025	40,000	28.1	14.0
43		0.1020	40,600	29.8	16.0
44		0.1025	40,500	29.8	16.0
	Average...........	0.1024	40,675	28.7	15.5
45	Copper Range copper reduced 22.5 per cent. by rolling.	0.1020	43,700	22.9	10.5
46		0.1020	42,900	21.9	10.0
47		0.1020	42,800	22.5	9.5
48		0.1020	43,000	23.5	11.0
	Average...........	0.1020	43,100	22.7	10.3

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* Calculated from average thickness of strips and average thickness before rolling.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips (Continued)*

Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.

No.	Brand of Copper and Reduction by Rolling*	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
49	Electrolytic copper reduced 38.2 per cent. by rolling.	0.0810	47,900	10.0	5.0
50		0.0810	47,700	11.0	5.0
51		0.0810	47,900	7.5	5.0
52		0.0810	47,500	8.7	5.0
	Average...........	0.0810	47,750	9.3	5.0
53	Mohawk copper reduced 37.2 per cent. by rolling.	0.0810	49,100	11.5	6.0
54		0.0805	49,500	10.2	5.0
55		0.0810	49,600	9.4	5.0
56		0.0805	49,600	9.4	5.0
	Average...........	0.0807	49,450	10.1	5.2
57	Copper Range copper reduced 38.1 per cent. by rolling.	0.0815	50,500	12.1	5.0
58		0.0815	49,800	7.2	5.0
59		0.0815	50,400	7.3	4.0
60		0.0815	50,500	6.2	5.0
	Average...........	0.0815	50,300	8.2	4.8
61	Electrolytic copper reduced 50.8 per cent. by rolling.	0.0645	49,900	4.2	3.0
62		0.0645	52,400	5.0	3.0
63		0.0645	52,200	5.9	3.5
64		0.0645	53,100	4.9	3.5
	Average...........	0.0645	51,900	5.0	3.2
65	Mohawk copper reduced 49.8 per cent. by rolling.	0.0645	53,800	3.0	3.5
66		0.0645	52,400	4.4	3.0
67		0.0645	53,500	5.0	3.0
68		0.0645	52,600	5.2	3.0
	Average...........	0.0645	53,325	4.4	3.1
69	Copper Range copper reduced 51.4 per cent. by rolling.	0.0640	54,700	2.8	4.0
70		0.0640	54,000	3.6	4.0
71		0.0640	54,900	4.3	4.0
72		0.0640	55,200	4.3	4.0
	Average...........	0.0640	54,700	3.7	4.0
73	Electrolytic copper reduced 60.8 per cent. by rolling.	0.0510	54,900	3.9	3.5
74		0.0520	55,000	2.4	2.5
75		0.0510	51,000	4.8	2.5
76		0.0515	56,400	3.2	3.5
	Average...........	0.0514	54,325	3.6	3.0
77	Mohawk copper reduced 60.7 per cent. by rolling.	0.0505	57,000	2.1	3.0
78		0.0505	57,400	2.1	2.5
79		0.0505	57,000	2.1	3.0
80		0.0505	56,800	2.1	3.0
	Average...........	0.0505	57,050	2.1	2.9
81	Copper Range copper reduced 61.2 per cent. by rolling.	0.0510	57,600	3.9	3.0
82		0.0510	58,100	3.9	3.0
83		0.0510	57,900	3.4	3.0
84		0.0510	57,400	4.0	3.0
	Average...........	0.0510	57,750	3.8	3.0
85	Electrolytic copper reduced 69.1 per cent. by rolling.	0.0410	57,000	3.3	3.0
86		0.0400	57,400	3.7	3.5
87		0.0405	57,700	2.3	3.0
	Average...........	0.0405	57,367	3.1	3.2
88	Mohawk copper reduced 68.1 per cent. by rolling.	0.0405	57,800	2.6	2.0
89		0.0410	58,400	2.6	2.0
90		0.0410	59,000	2.6	2.0
91		0.0410	59,000	2.6	2.0
	Average...........	0.0409	58,550	2.6	2.0
92	Copper Range copper reduced 70.0 per cent. by rolling.	0.0395	58,800	2.1	1.5
93		0.0395	59,300	1.4	2.0
94		0.0395	60,500	3.7	2.5
95		0.0395	61,200	2.4	2.5
96		0.0395	60,500	2.4	2.0
	Average...........	0.0395	60,060	2.4	2.1

*Calculated from average thickness of strips and average thickness before rolling.

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling*[*]

Average thickness before annealing, 0.064 to 0.067 in.

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
97	Electrolytic copper, unannealed (as received from the mill).	0.0640	51,600	4.56	4.0
98		0.0645	51,600	3.20	4.0
99		0.0670	51,600	4.57	4.0
100		0.0640	51,600	4.56	4.0
	Average...........	0.0649	51,600	4.22	4.0
101	Mohawk copper, unannealed (as received from the mill).	0.0670	52,200	4.94	4.0
102		0.0665	52,100	5.03	4.0
103		0.0665	52,100	6.02	4.0
104		0.0660	52,100	4.66	4.0
	Average...........	0.0665	52,125	5.16	4.0
105	Copper Range copper, unannealed (as received from the mill).	0.0665	53,000	8.83	4.0
106		0.0665	53,500	9.00	4.5
107		0.0665	53,000	8.83	4.5
108		0.0665	52,900	8.60	4.5
	Average...........	0.0665	53,100	8.81	4.4
109	Electrolytic copper annealed at 100°C.	0.0640	51,700	3.74	4.0
110		0.0640	51,600	4.56	4.0
111		0.0640	51,400	4.56	4.0
112		0.0640	51,400	3.96	4.0
	Average...........	0.0640	51,525	4.21	4.0
113	Mohawk copper, annealed at 100°C.	0.0660	52,300	4.63	4.0
114		0.0655	52,900	6.86	5.0
115		0.0660	52,400	5.01	4.0
116		0.0660	52,300	4.63	4.0
	Average...........	0.0659	52,475	5.28	4.2
117	Copper Range copper, annealed at 100°C.	0.0660	52,500	8.85	6.0
118		0.0660	52,500	8.85	6.0
119		0.0660	53,300	7.65	4.5
120		0.0640	54,400	4.77	4.0
	Average...........	0.0655	53,175	7.53	5.1
121	Electrolytic copper, annealed at 200°C.	0.0650	49,000	9.86	4.5
122		0.0645	49,100	9.49	4.5
123		0.0650	48,600	6.12	4.0
124		0.0645	50,500	5.15	6.0
	Average...........	0.0647	49,300	7.66	4.8
125	Mohawk copper, annealed at 200°C.	0.0660	52,900	9.45	6.5
126		0.0660	52,000	8.85	5.0
127		0.0660	52,100	7.45	5.5
128		0.0660	52,000	7.45	5.5
	Average...........	0.0660	52,250	8.30	5.6
129	Copper Range copper, annealed at 200°C.	0.0640	53,100	5.81	5.5
130		0.0655	52,500	12.95	4.5
131		0.0650	52,700	12.85	4.5
132		0.0660	51,800	13.13	4.5
	Average...........	0.0651	52,525	11.18	4.8

[*] Calculated values of reduction vary between 49 and 51 per cent.

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling* (*Continued*)

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
133	Electrolytic copper, annealed at 250°C.	0.0645	49,200	20.80	9.0
134		0.0645	49,100	19.30	9.0
135		0.0650	48,500	19.05	8.5
136		0.0645	48,900	18.30	9.0
	Average...........	0.0646	48,925	19.36	8.9
137	Mohawk copper, annealed at 250°C.	0.0657	51,800	15.30	7.5
138		0.0650	51,800	14.85	7.5
139		0.0655	51,700	14.00	7.5
140		0.0652	51,700	14.25	8.0
	Average...........	0.0654	51,750	14.60	7.6
141	Copper Range copper, annealed at 250°C.	0.0655	52,500	12.00	6.0
142		0.0660	51,500	14.15	8.0
143		0.0660	53,000	14.15	8.0
144		0.0660	52,600	14.45	7.5
	Average...........	0.0659	52,400	13.69	7.4
145	Electrolytic copper, annealed at 275°C.	0.0643	43,000	24.40	19.0
146		0.0650	43,000	24.60	19.0
147		0.0635	43,600	22.80	(14.0)
148		0.0650	42,500	24.60	20.0
	Average...........	0.0645	43,025	24.10	19.3
149	Mohawk copper, annealed at 275°C.	0.0650	51,200	16.65	8.5
150		0.0658	50,700	15.65	8.5
151		0.0655	50,700	15.40	7.0
152		0.0657	50,700	16.00	8.5
	Average...........	0.0655	50,825	15.92	8.1
153	Copper Range copper, annealed at 275°C.	0.0653	52,200	(9.45)	(5.0)
154		0.0655	51,500	11.00	9.0
155		0.0655	51,000	11.00	9.0
156		0.0655	51,200	11.00	9.0
	Average...........	0.0654	51,475	11.00	9.0
157	Electrolytic copper, annealed at 300°C.	0.0640	38,800	36.90	36.5
158		0.0640	37,600	40.20	36.0
159		0.0650	35,400	(44.00)	(52.5)
160		0.0640	37,700	38.00	37.5
	Average...........	0.0647	37,375	38.37	36.7
161	Mohawk copper, annealed at 300°C.	0.0655	50,700	10.75	11.0
162		0.0660	50,400	15.65	11.0
163		0.0660	50,000	(8.85)	11.0
164		0.0660	50,100	14.10	11.0
	Average...........	0.0659	50,300	13.50	11.0
165	Copper Range copper, annealed at 300°C.	0.0660	50,200	14.50	9.5
166		0.0645	49,800	13.40	11.5
167		0.0660	51,000	14.35	10.0
168		0.0655	50,400	15.00	9.0
	Average...........	0.0655	50,350	14.31	10.0

TABLE 3.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures after a Reduction of* 50 *Per Cent. by Rolling* (*Continued*)

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
169	Electrolytic copper, annealed at 325°C.	0.0650	34,600	44.60	46.0
170		0.0650	35,100	43.90	46.0
171		0.0630	34,600	42.00	49.0
172		0.0650	35,600	43.10	(43.0)
173		0.0645	33,800	44.60	49.5
	Average	0.0645	34,740	43.64	47.6
174	Mohawk copper, annealed at 325°C.	0.0650	43,000	27.90	19.0
175		0.0645	43,500	28.40	19.0
176		0.0684	43,500	30.70	18.0
177		0.0650	41,800	29.60	22.0
	Average	0.0657	42,950	29.15	19.5
178	Copper Range copper, annealed at 325°C.	0.0655	37,600	39.90	42.0
179		0.0655	37,550	39.80	41.0
180		0.0655	36,800	40.20	43.0
181		0.0640	38,200	39.10	42.0
	Average	0.0651	37,540	39.75	42.0
182	Electrolytic copper, annealed at 350°C.	0.0650	33,900	45.90	50.0
183		0.0630	34,300	45.00	51.0
184		0.0645	34,100	46.30	52.0
185		0.0650	33,900	46.40	51.0
	Average	0.0644	34,050	45.90	51.0
186	Mohawk copper, annealed at 350°C.	0.0663	34,500	44.00	54.0
187		0.0650	34,200	45.40	54.0
188		0.0655	34,700	45.40	53.0
189		0.0655	34,700	43.30	54.0
	Average	0.0656	34,525	44.53	53.8
190	Copper Range copper, annealed at 350°C.	0.0650	34,900	43.00	54.0
191		0.0655	34,700	42.60	54.0
192		0.0660	35,100	43.00	54.0
193		0.0655	34,500	42.20	53.5
	Average	0.0655	34,800	42.70	53.9
194	Electrolytic copper, annealed at 400°C.	0.0640	34,100	44.60	51.0
195		0.0645	34,000	45.10	55.0
196		0.0645	33,900	45.00	54.0
197		0.0645	33,000	45.00	51.5
	Average	0.0644	33,750	44.93	52.9
198	Mohawk copper, annealed at 400°C.	0.0660	34,400	44.50	53.5
199		0.0660	34,400	45.00	53.5
200		0.0660	34,100	45.50	53.5
201		0.0660	34,500	44.90	54.0
	Average	0.0660	34,350	44.98	53.6
202	Copper Range copper, annealed at 400°C.	0.0655	34,700	47.60	54.0
203		0.0655	34,600	46.50	53.5
204		0.0645	34,200	47.10	54.0
205		0.0657	34,400	46.10	54.5
	Average	0.0650	34,475	46.83	54.0

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
206	Electrolytic copper, annealed at 500°C.	0.0635	33,500	45.10	57.5
207		0.0635	33,500	46.50	56.0
208		0.0635	33,200	44.50	49.0
209		0.0640	33,500	46.00	57.0
	Average...........	0.0636	33,425	45.53	54.9
210	Mohawk copper, annealed at 500°C.	0.0640	33,200	46.80	49.5
211		0.0655	33,900	49.30	57.0
212		0.0660	33,500	49.50	56.5
213		0.0650	32,800	49.60	55.0
	Average...........	0.0651	33,350	48.80	54.8
214	Copper Range copper, annealed at 500°C.	0.0655	33,500	47.90	55.0
215		0.0645	33,400	49.00	58.0
216		0.0645	33,700	45.50	54.5
217		0.0645	33,500	48.50	58.0
	Average...........	0.0648	33,525	47.73	56.4
218	Electrolytic copper, annealed at 600°C.	0.0645	33,400	46.90	54.5
219		0.0640	33,300	46.10	54.5
220		0.0640	33,300	46.40	55.0
221		0.0620	33,800	45.40	55.0
	Average...........	0.0636	33,450	46.20	54.8
222	Mohawk copper, annealed at 600°C.	0.0645	33,200	48.40	55.5
223		0.0645	33,100	47.00	54.0
224		0.0645	32,900	49.60	56.0
225		0.0655	33,000	49.40	56.5
	Average...........	0.0648	33,050	48.60	55.5
226	Copper Range copper, annealed at 600°C.	0.0655	33,100	49.40	57.0
227		0.0655	33,000	48.60	57.0
228		0.0660	33,250	51.50	57.5
229		0.0655	33,200	51.10	56.0
	Average...........	0.0656	33,140	50.15	56.9
230	Electrolytic copper, annealed at 700°C.	0.0635	32,200	44.70	52.5
231		0.0630	31,800	44.90	55.5
232		0.0640	32,199	45.00	55.0
233		0.0635	32,400	46.40	55.0
	Average...........	0.0635	32,125	45.25	54.5
234	Mohawk copper, annealed at 700°C.	0.0645	31,900	49.50	54.0
235		0.0647	32,200	47.90	61.0
236		0.0645	32,200	49.30	55.0
237		0.0645	32,100	49.20	53.5
	Average...........	0.0646	32,050	48.98	55.9
238	Copper Range copper, annealed at 700°C.	0.0650	32,300	50.60	55.5
239		0.0645	32,100	50.10	59.0
240		0.0643	32,000	49.70	55.0
241		0.0645	31,900	49.70	57.0
	Average...........	0.0646	32,075	50.03	56.6

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling* (*Continued*)

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
242	Electrolytic copper, annealed at 800°C.	0.0615	32,400	44.00	49.0
243		0.0615	32,200	44.30	51.0
244		0.0600	32,100	42.80	51.0
245		0.0615	32,500	44.00	50.5
	Average...........	0.0615	32,300	43.78	50.4
246	Mohawk copper, annealed at 800°C.	0.0615	32,100	44.80	52.0
247		0.0620	32,200	45.70	53.0
248		0.0630	31,800	45.80	53.0
249		0.0630	31,400	46.30	51.0
	Average...........	0.0624	31,875	45.65	52.3
250	Copper Range copper, annealed at 800°C.	0.0620	32,200	46.60	55.0
251		0.0625	32,100	46.00	55.0
252		0.0625	32,100	47.40	59.0
253		0.0613	31,400	47.10	50.0
	Average...........	0.0621	31,950	46.79	54.8
254	Electrolytic copper, annealed at 900°C.	0.0600	31,600	39.80	44.0
255		0.0615	31,200	41.40	44.0
256		0.0620	32,200	41.10	46.0
257		0.0600	31,800	39.80	44.0
	Average...........	0.0609	31,700	40.53	44.5
258	Mohawk copper, annealed at 900°C.	0.0640	30,100	46.90	47.0
259		0.0620	30,600	42.40	45.0
260		0.0605	30,600	44.60	46.0
261		0.0620	31,500	45.80	44.0
	Average...........	0.0621	30,700	44.93	45.5
262	Copper Range copper, annealed at 900°C.	0.0625	30,200	48.50	54.5
263		0.0620	30,200	47.50	51.0
264		0.0610	30,500	45.60	45.0
265		0.0610	30,900	46.50	48.0
	Average...........	0.0616	30,450	47.03	49.6
266	Electrolytic copper, annealed at 1,000°C.	0.0600	29,700	37.16	35.5
267		0.0600	30,500	32.10	31.0
268		0.0565	30,900	33.30	30.0
269		0.0570	32,300	31.90	42.5
	Average...........	0.0589	30,850	33.62	34.8
270	Mohawk copper, annealed at 1,000°C.	0.0600	30,400	40.40	43.0
271		0.0600	29,800	40.40	38.0
272		0.0605	29,900	40.00	37.0
273		0.0590	30,200	39.40	36.0
	Average...........	0.0599	30,075	40.05	38.5
274	Copper Range copper, annealed at 1,000°C.	0.0610	28,800	40.90	35.0
275		0.0580	29,700	38.82	37.0
276		0.0585	29,100	38.60	36.0
277		0.0605	27,600	41.50	37.0
	Average...........	0.0595	28,800	39.96	36.3

(2) *Tests after Anneal under Oxidizing Conditions. Second Set of Strips.*

Table 3 shows the results of tests made on strips annealed 40 min. at temperatures ranging from 100° to 1,000°C. The previous history of the strips is given in the sketches of Fig. 1. The methods of annealing are described in the preceding section. Preheating periods are given in Table 1.

While all strips were annealed at gage No. 14 (B. & S.), the thickness is somewhat reduced by oxidation after anneal at high temperatures and, in order that comparisons of this sort may be made, micrometer measurements of thickness are given in the table.

(3) *Tests after Anneal under Reducing Conditions. Second Set of Strips*

Table 4 shows the results of a few tests made on strips annealed 40 min. at 600°, 800°, and 1,000° C., respectively, in an atmosphere of illumi-

TABLE 4.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures in an Atmosphere of Coal Gas**

Average thickness, before annealing, 0.064 to 0.067 in.

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Ultimate Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
278	Electrolytic copper, an-	0.0690	29,400	Broke at shoulder	(22.0)
279	nealed at 600° C.	0.0680	29,100	Broke at shoulder	(22.0)
	Average..........	0.0685	29,250	....	(22.0)
280	Mohawk copper, an-	0.0690	29,500	Broke at shoulder	(30.0)
281	nealed at 600° C.	0.0670	30,500	Broke at shoulder	(25.5)
	Average..........	0.0680	30,000	....	(27.8)
282	Copper Range copper,	0.0670	31,800	39.20 Good breaks	41.0
283	annealed at 600° C.	0.0670	31,700	39.40 Good breaks	43.0
	Average..........	0.0670	31,750	39.30 Good breaks	42.0
284	Electrolytic copper, an-	0.0710	21,500	12.35 Good break	12.0
285	nealed at 800° C.	0.0710	17,700	Broke in several	(9.0)
	Average..........	0.0710	19,600	pieces	(10.5)
286	Mohawk copper, an-	0.0700	17,600	Broke at shoulder	(8.0)
287	nealed at 800° C.	0.0690	19,000	Broke in jaws	....
	Average..........	0.0695	18,300		
288	Copper Range copper,	0.0695	20,700	12.80 Good breaks	11.0
289	annealed at 800° C.	0.0700	21,700	Broke at shoulder	(11.0)
	Average..........	0.0698	21,200	....	11.0
290	Electrolytic copper, an-	0.0670	21,900	Broke at shoulder	(14.5)
291	nealed at 1,000° C.	0.0710	22,300	Broke at shoulder	(14.5)
	Average..........	0.0690	22,100	....	(14.5)
292	Mohawk copper, an-	0.0690	21,300	Broke in pieces	(16.5)
293	nealed at 1,000° C.	0.0700	21,700	18.15 Good breaks	17.0
	Average..........	0.0695	21,500	....	(16.8)
294	Copper Range copper,	0.0690	22,100	Broke in jaws	.....
295	annealed at 1,000° C.	0.0690	22,100	16.70 Good breaks	16.0
	Average..........	0.0690	22,100	....	......

* Calculated values of reduction vary between 49 and 51 per cent.

nating gas. The previous history and dimensions of the strips are the same as in the case of the preceding tests.

Discussion of Tables, Curves, and Microstructure

(1) *Tests after Cold-Rolling*

The results assembled in Table 2 have been plotted in Figs. 5, 6, and 7. In Fig. 5, the comparative values of tensile strength in the three kinds of metal are shown; in Fig. 6, the comparative values of elongation; and in Fig. 7, the comparative values of reduction of area.

It will be observed that changes in temper of about the same degree are produced by rolling the three kinds of metal through any given value of reduction. In other words, these tests fail to develop any striking differences of rolling temper between the different brands of copper used. The difficulty of obtaining a high degree of accuracy in the ordinary tensile test is a handicap in comparing products which are very similar in nature. Testing discrepancies of several per cent. will ordinarily occur with careful handling of test strips from the same sheet of metal. The uneven distribution of small percentages of impurity in commercial products is a source of irregularities in this respect. As may be seen in the micrographs (Fig. *d*, Plate 4, is typical of this condition), the oxide particles, originally more or less segregated in the cast cake, are drawn out into continuous trains in the rolled product and it is obvious that there will be a closer grouping of the zones, or attenuated lines of weakness in some cross-sectional areas than in others. Furthermore, the presence of oxide interferes with the normal growth of grain on annealing, and, consequently, there will be irregularities of grain size in the copper matrix corresponding to segregations of oxide. Such a condition is shown admirably in Fig. *a*, Plate 4, where coarser grain has formed on both sides of an extended zone or band sprinkled with oxide particles.[9] Such conditions cannot fail to be responsible for variations in breaking load, elongation, and reduction of area among the duplicate test strips.

An accurate average (representing the mean properties of the metal) could, of course, be struck from tests on a large number of strips. There is always a limit in this direction, however, and we decided upon four strips as the largest number which could be handled in this investigation. It will be noted from examination of the tabulated results that considerable variations both of strength and elongation were obtained in a given set of tests. It may be left to the individual reader to form his own opinion as to the actual value of the averages reported.

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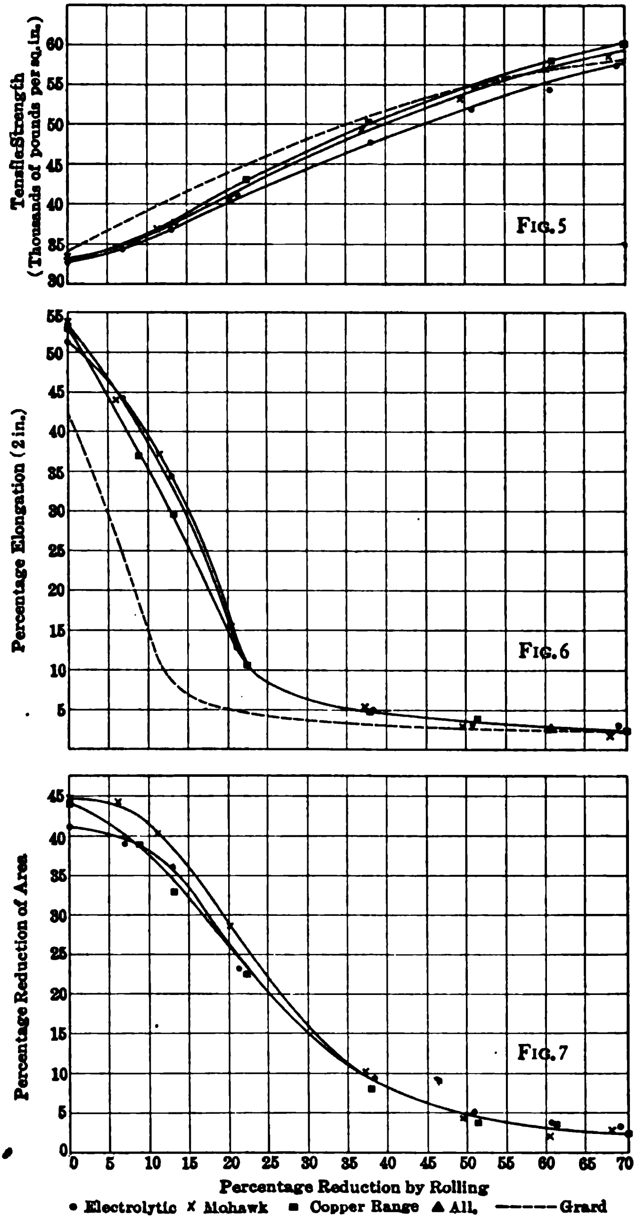
[9] Mathewson and Caesar, in a paper lately sent to the *Internationale Zeitschrift für Metallographie*, have discussed the effect of oxide on the size of recrystallized grain in pure copper.

In rejecting any seemingly abnormal result, we endeavored to be perfectly fair to each kind of metal. In general, where serious irregularities developed, an entirely new set of bars was tested and all of the earlier results rejected; in no case were four especially favorable or unfavorable



FIGS. 5, 6, AND 7.—STRENGTH AND DUCTILITY OF ROLLED STRIPS.

results selected from a long series of tests, but, in some cases, averages were struck from three tests when there seemed good reason to reject the fourth.

Such differences between the three varieties of copper as appear to be real, although small, may be summarized in the following statements:

The Copper Range product decreases in ductility as measured by elongation more rapidly than the electrolytic product during the early stages of rolling (down to about 20 per cent. reduction of area by rolling), but both products increase in strength about equally, while, during the later stages of rolling, both products decrease about equally in ductility and the Copper Range metal increases in strength more rapidly than the electrolytic metal. The Mohawk copper occupies a more or less intermediate position.

Differences in ductility as measured by reduction of area are also confined to the earlier range of reductions by rolling. While, in principle, the measurement of reduction of area possesses some advantages over the measurement of elongation as a criterion of deformational properties, since the former is more nearly independent of the shape of the test piece, it is far more difficult to make in the case of thin flat metal. Our measurements of elongation are, therefore, more reliable than the corresponding measurements of reduction of area. In the present case, all three kinds of material show practically the same values of reduction of area after the reduction by rolling has passed 35 per cent.

From the general point of view that the Copper Range product takes a harder temper on rolling than the electrolytic product, but does not ultimately become less ductile, it would seem that both might be able to stand the same amount of punishment by rolling, but that the latter might roll a little easier. Since the pressure exerted in rolling cannot be calculated from the results of the ordinary tensile test, and since it would require, in any event, comparative determinations of the yield-point stress intensities at intervals throughout the rolling range in order to determine which of the two materials would require the greatest load to start permanent deformation at any stage, the above statement must remain subject to eventual qualification by actual test.

Where it is desired to finish a previously annealed product with a limited amount of ductility, but increased strength, by rolling through several numbers of gage, it is quite evident that an appreciable increase in strength without sacrifice of ductility may be secured by using Copper Range metal instead of electrolytic metal. Thus, rolling from No. 8 to No. 12, B. & S. gage, a reduction of about 40 per cent., would give the Copper Range material a tensile strength about 1 ton (per square inch) greater than that of the electrolytic material similarly treated, while the elongation in both cases would be in the neighborhood of 5 per cent.

In order that the well-known curves of Grard[10] (electrolytic copper) may be compared with our own, they have been dotted into Figs. 5 and 6. Grard's measurements of elongation were made on a test section 100 mm. long (approximately 4 in.) and, for this reason, show much lower values than our own, measured in 2 in., until the region of slight ductility is reached at about a 35 per cent. reduction by rolling. From this point on, the stretch becomes sensibly uniform along the entire test length, making it of little consequence whether a length of 2 or 4 in. is chosen, and our results are practically coincident with Grard's.

Grard's curve of breaking strength starts somewhat above and ends somewhat below our curves. Thus, our curves show a higher rate of increase of strength with the degree of reduction by rolling. This is doubtless due to differences in the dimensions of the test strips. For reasons previously given, we used test strips whose thickness decreased progressively with the increase in reduction by rolling (minimum ratio of test length to $\sqrt{\text{sectional area}}$, 6½ to 1; maximum ratio, 11½ to 1), while Grard's practice in this respect is not clear. He shows a ratio of 5 to 1 in a sketch of the test piece, but later states that a variable final cross-section was used.

(2) *Tests after Annealing under Oxidizing Conditions*

The results assembled in Table 3 have been plotted in Figs. 8 (strength), 9 (elongation), and 10 (reduction of area). For general purposes of comparison, Grard's[11] curves from tests on 2-mm. strips of electrolytic copper and Bardwell's[12] curves from tests on No. 12, B. & S. wire, containing from 0.036 to 0.070 per cent. of oxygen and an indeterminate amount of silver, are shown in dotted form.

Naturally, one's attention is first drawn to the difference between electrolytic copper, on the one hand, and the arsenical coppers, on the other hand, with respect to the earliest manifestation of an annealing effect. The temperature range of early transformation (annealing period of 40 min.) from the strain-hardened condition to the recrystallized, or annealed, condition is equally narrow in all three kinds of material; it is practically coincident in the two arsenical coppers, but lies some 50° lower in the electrolytic copper.

It is believed that these are the first systematic tests designed to reveal the relative softening (annealing) characteristics of commercially pure copper and commercial copper containing arsenic. The general tendency of arsenic to raise the effective annealing temperature has long

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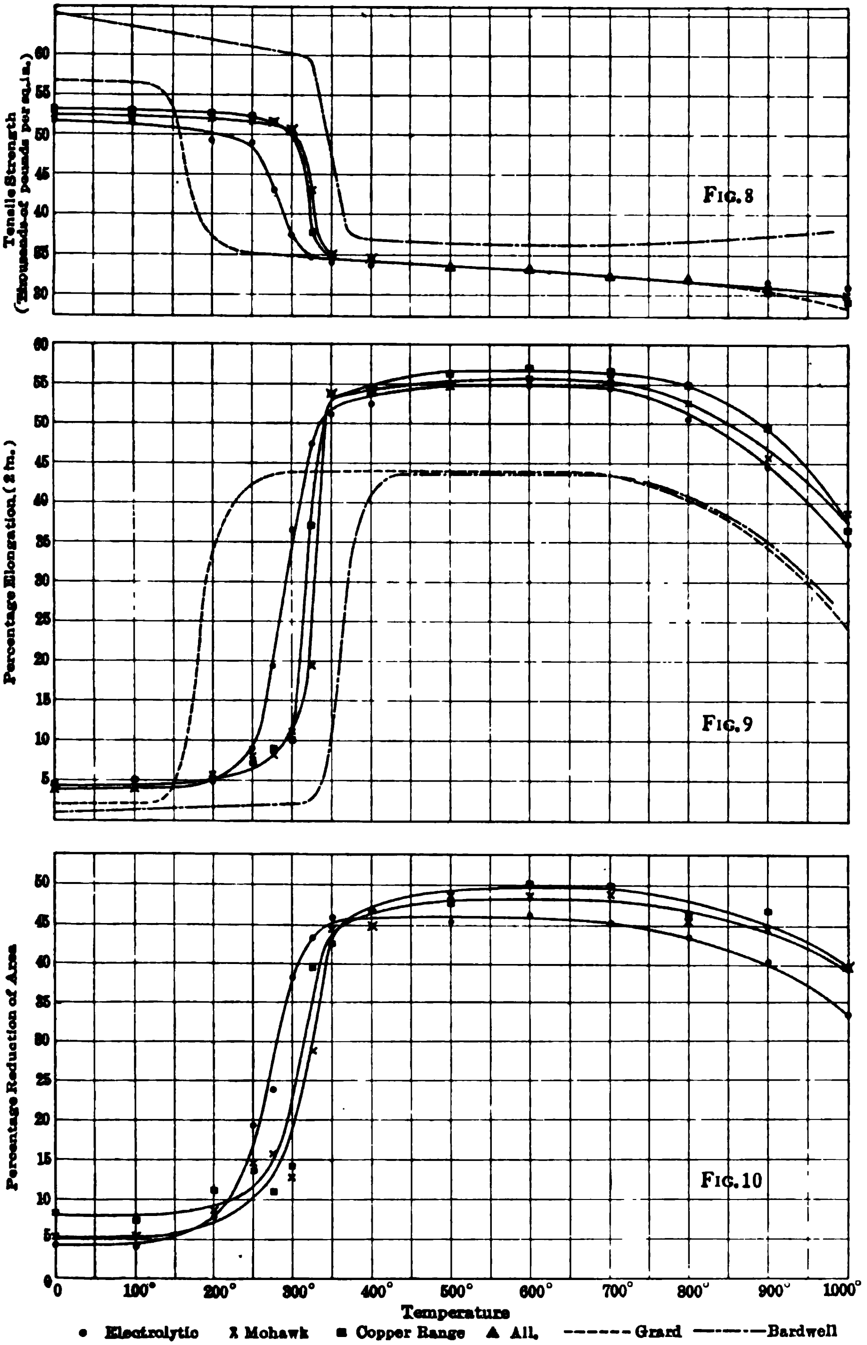
[10] *Loc. cit.*

[11] *Loc. cit.*

[12] E. S. Bardwell: The Annealing of Cold-Rolled Copper, *Trans.*, vol. 49, pp. 753–773 (1914).



Figs. 8, 9, and 10.—Strength and Ductility of Annealed Strips.

been known on the basis of fragmentary tests, here and there, under a variety of conditions difficult to bring into any sort of alignment. Thus, F. Johnson[13] quotes from Tomlinson to the effect that samples of rolled arsenical copper containing 0.36 per cent. of arsenic did not soften after 3 or 4 weeks at a temperature approaching 400° F. (205° C.), while samples of pure copper were practically annealed. Further, the arsenical copper lost only 6 per cent. of its strength after 4 hr. at 870° C., while the pure copper lost 44 per cent. (*i.e.*, was presumably completely annealed).

The first statement above is quite in harmony with the conditions shown by our tests, viz., the range of softening by recrystallization from the strain-hardened condition is always carried to lower temperatures by prolonging the anneal. In softening, the arsenical metal lags behind the pure copper (temperature-lag of 50° C. for an annealing period of 40 min. in the case of metal containing 0.3 per cent. of arsenic, neglecting the possible effect of silver) and the above result signifies that the stated temperature of 205° C. lies well within the temperature range of softening based upon an annealing period of several weeks in the case of pure copper, while, in the case of the arsenical metal, it lies below, or barely within, the temperature range of softening based upon a similar annealing period.

It is perfectly clear, without undertaking special experiments involving variation of the annealing period, that arsenical coppers, such as the Copper Range product, constitute a commonly available and satisfactory material—superior to pure copper—when it is desired to preserve a rolled temper in the face of considerable variation of temperature up to some 200° C.

The second statement taken from Mr. Johnson's paper, which seems to indicate a difficulty in annealing the arsenical metal, cannot be brought into harmony with the results of our tests. We find that the separate temperature ranges of early softening for the three kinds of material end at a very nearly common (minimum) temperature of complete softening (some 325° C.) from which point onward, there are only minor differences in the annealing characteristics of the different kinds of metal. It is inconceivable that any of these products should lose only 6 per cent. of their strength by annealing for a period of hours at red heat. Such an effect must be attributed to the somewhat higher arsenic content (of doubtful application), or to other unknown peculiarities of composition.

As far as the present comparisons go, we are able to state that the proper annealing temperature for electrolytic copper (dull red) will produce equally satisfactory results in the case of Copper Range or Mohawk copper.

Outstanding features of comparison between the three varieties of copper in the annealed condition may be summarized as follows:

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[13] F. Johnson: The Influence of Impurities on the Properties of Copper, *Metallurgical and Chemical Engineering*, vol. 8, pp. 570–575 (October, 1910).

Only minor differences in strength throughout the entire annealing range of 400 to 1,000° C. were observed. As a rule, after a given anneal, the difference between extreme values of tensile strength (averages) was less than 1,000 lb. per square inch.

The Copper Range product consistently showed the highest values of elongation and reduction of area and the electrolytic product the lowest, with the Mohawk product occupying a more or less intermediate position.

It is important to observe that, in the temperature region of deterioration, which begins in the neighborhood of 700° with all three varieties, the relative ductilities of the Copper Range and electrolytic products are such that the former will stand an extra 100°, or so, of overheating before it drops to the ductility value of the latter. This means that if the metal is allowed to come to a bright red heat in annealing, a Copper Range load may be expected to emerge with about the same ductility as an electrolytic load annealed at the proper temperature of low redness, while an electrolytic load under the same conditions would presumably fall below this standard.

It will be noticed that Grard's curve of tensile strength practically coincides with our curves after an annealing temperature of about 300° is reached, while Bardwell's curve lies somewhat above ours, as would be expected in the case of wires, but rises in the region of high temperature anneals for some obscure reason. The tests from these different sources show marked discrepancies with respect to the temperature ranges of initial softening. Since the location of this range (*zone de detente* in Grard's phraseology) depends upon the period of anneal and the degree of preliminary cold-working, certain discrepancies would be expected, although hardly of the order shown in these comparisons. As may be seen from the initial strength values, both Grard and Bardwell started their tests with material which had been more severely reduced (by rolling or drawing) than our material. Making allowance for the higher strengths commonly shown by wires, it appears that the materials used by both of these authors had received approximately the same degree of cold-working and, under similar annealing conditions, both should show a lower range of initial softening than our material. Bardwell kept his wires 20 min. in the furnace and Grard kept his strips 30 min. in the furnace, as far as we can determine from his general description of procedure. Our material was reduced 50 per cent. by rolling and we used an annealing period of 40 min., actual time of heating at the annealing temperature.

We do not know how far the presence of a small amount of silver might have effected Bardwell's results, but, on a basis of equivalent purity, the temperature ranges of initial softening found by both authors should not be greatly dissimilar and they should lie somewhat behind the corresponding range found by us in the case of electrolytic copper. As matters stand, the range determined in our experiments occupies an intermediate position. Not enough definite information is at present available to permit an harmonious interpretation of these irregularites. It is particularly to be desired that determinations of the equilibrium positions of the temperature range of initial softening of rolled electrolytic copper for various reductions be made.

Our curves of elongation similarly occupy an intermediate position between those of Bardwell and Grard. Owing to the divergent ratios of test length to diameter (or to $\sqrt{\text{sectional area}}$), the actual values of elongation vary within wide limits. A purely fortuitous coincidence between Grard's and Bardwell's values is noted at annealing temperatures above 400°. This is due to the fact that Grard used a ratio of approximately 22 to 1 (length to $\sqrt{\text{sectional area}}$) and Bardwell, a ratio of approximately 150 to 1 (length to diameter), which choice of dimensions appears to have compensated for the flatter gradient of stretch of the wires, yielding the same percentage values of elongation.

All of the curves show practically the same temperature range of deterioration (*zone de fléchissement*, in Grard's phraseology), viz., from 700° upward.

(3) *Tests after Annealing under Reducing Conditions*

Bengough and Hill[14] obtained some strikingly destructive effects by annealing certain arsenical mixtures in reducing gases, which led them to seek for an explanation in the constitutional relationships between copper, arsenic, and oxygen. Further tests by the same authors and also by Johnson[15] and others have established the fact that both arsenical and non-arsenical coppers containing oxygen suffer deterioration by "gassing" when given a reducing anneal above moderate red heat, while deoxidized copper is not subject to this disadvantage. It is thus reasonably certain that the effect in question is due to an ordinary reduction of cuprous oxide by hydrogen or other gases which can permeate (*i.e.* dissolve in) solid copper, with formation of (bulky) gaseous products, thereby causing fissures and cavities to develop.

We have endeavored to ascertain the relative destructive effects which occur on annealing the three kinds of copper in hand under typical reducing conditions at several representative temperatures. Accordingly, the test strips were annealed at 600°, 800°, and 1,000° C. in an iron tube through which coal gas was passed in sufficient quantity to burn freely at the discharge end, notwithstanding a certain amount of decomposition, with formation of soot, in the tube.

On examination of Table 4, in which the results of these tests are to be found, it will be seen that, of the whole lot of strips tested, only one set, viz., that composed of strips from the Copper Range material annealed at 600°, showed a fair retention of ductility. Here, an elongation of 42 per cent. and a reduction of area of 39.3 per cent. replace the normal values of 56.9 and 50.15, respectively, attained after an oxidizing anneal at the same temperature. The other materials after anneal at this temperature gave unsatisfactory breaks and lower values of ductility, as far as could be ascertained. Destructive effects—brittleness and an almost entire lack of elongation—were obtained throughout in the case of reducing anneals at 800° and 1,000°. As far as may be judged from the small number of tests in which good breaks were secured, the anneals at 1,000° were somewhat less destructive than the anneals at 800°. This was doubtless due to the protective influence of a dense coating of carbonaceous material which was deposited on the strips at the highest annealing temperature.

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[14] *Loc. cit.*

[15] F. Johnson: A Method of Improving the Quality of Arsenical Copper, *Journal of the Institute of Metals*, vol. 10, pp. 283–4 (1913).

All fractures showed a lack of luster and an approximation to the brick-red color of fractures from "set" copper.

It seemed of considerable interest to determine the oxygen content of the "gassed" strips in the usual way, viz., by ignition in hydrogen. Accordingly, representative samples were sent to the laboratory of the Michigan Smelting Co. for this purpose. Table 5 embodies the results of analyses from this source.

TABLE 5.—*Oxygen Content of Annealed Copper*

No. of Test Strip (See Table 4)	Kind of Copper	Temperature of Anneal, Degrees Centigrade	Oxygen Content,* Per Cent.
278	Electrolytic	600	0.0456
280	Mohawk	600	0.0441
282	Copper Range	600	0.0402
284	Electrolytic	800	0.0035
286	Mohawk	800	0.0060
288	Copper Range	800	0.0082
290	Electrolytic	1,000	0.0054
293	Mohawk	1,000	0.0010
294	Copper Range	1,000	0.0053

In view of the small percentages of oxygen found in the material annealed at 800° and 1,000°, it is obviously not justifiable to attach great importance to the differences which may be observed in comparing the several results. The analytical data are in general agreement with the testing results. In particular, the figures indicate partial deoxidation of the material at 600° and almost complete deoxidation at 800° and 1,000°.

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* The initial oxygen content of the different kinds of copper was as follows: Electrolytic, 0.071 per cent.; Mohawk, 0.052 per cent.; Copper Range, 0.055 per cent.

In general, the tests in this section indicate that the Copper Range material is somewhat less susceptible to the evil effects of a reducing anneal than either of the other varieties of copper. On the other hand, they show that none of these coppers will retain its ductility unimpaired after a 40-min. period of anneal under actively reducing conditions at a temperature even as low as 600°.

It is clearly of great importance to regulate carefully the combustion in all furnaces used for heating copper preparatory to hot-rolling, or for annealing copper after cold-working (unless the metal has previously been deoxidized).

Microstructure

Pieces were detached from the ends of the test strips, representing each separate anneal, for microscopic examination. Photomicrographs are reproduced in the accompanying plates in sets of three (electrolytic, Mohawk, and Copper Range copper, respectively, to correspond with the separate anneals at intervals of 100°, beginning with 200° and ending with 1,000°.

Figs. *a*, *b*, and *c*, of Plate 1, taken from strips of electrolytic, Mohawk, and Copper Range copper, respectively, annealed at 200°, are thoroughly representative of the structure shown by the cold-worked metal and by metal annealed at any temperature short of the softening point. The ordinary characteristics of severe deformation, viz., elongation of the grains in the direction of rolling and wavy lines or bands of deformation at right angles to the direction of rolling, are plainly evident in these photomicrographs.

The curves of tensile strength, elongation, and reduction of area (Figs. 8, 9, and 10, respectively) call for far-reaching recrystallization of the electrolytic copper and hardly more than incipient recrystallization of the other varieties at the next temperature of anneal, 300°. Accordingly, Fig. *d*, Plate 1, shows the presence of numerous minute secondary grains along with unaltered fragments of the primary grains—some of them still marked by lines of deformation, while Figs. *e* and *f* show very little evidence of recrystallization at this magnification (×77).[16]

Complete recrystallization has occurred in all cases at 400° and Plates 2 to 5, inclusive, show the growth of grain which takes place as the temperature of anneal is increased by increments of 100° between 400° and 1,000°.

Baucke[17] has published counting data which indicate that grain growth on annealing strain-hardened copper (of unknown purity) increases gradually up to 700°, but much more rapidly in the temperature region, 700 to 1,000° C.

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[16] A satisfactory examination for the early appearance of recrystallization can only be made at moderately high magnifications. For the sake of uniformity, however, we have prepared all photomicrographs at the magnification which has proved most serviceable in the general work.

[17] H. Baucke: On Some Recent Micrographical Investigations of Copper, *International Association for Testing Materials, 6th Congress (New York)*, Section II_{14}, vol. 2.

The photomicrographs of the present set show conditions quite in harmony with the above generalization which may thus be extended to include the three varieties of copper used in these tests.

We have not undertaken to prepare counting data from these photomicrographs, believing that the principal elements of difference are such as to render qualitative comparisons more valuable than attempted quantitative ones. To take the extreme cases, it is difficult to compare the grain size of the electrolytic samples with that of the Copper Range samples, because of greater variations in the former material. These major variations are due to the presence of finely subdivided oxide particles which are concentrated in certain portions of the copper matrix. In such regions, the mere presence of the oxide particles hinders coalescence between adjoining grains and reduces the final grain size.[18]

The form and distribution of these oxide segregations, as well as the relative conditions presented by the two kinds of copper, are best discussed with reference to their genesis in the cast material.

Fig. *b*, Plate 6, shows at the left a section through primary dendrites of copper in a dark ground mass of eutectic. Planimetric analysis shows an oxygen content of 0.14 per cent.; equivalent to 1.26 per cent. of cuprous oxide. From the regularity of arrangement, it is probable that most of the light areas are parts of the same dendritic growth, *i.e.*, that they are not separate grains, but are sections from the same grain with a eutectic filling in the interstices.

If we increase the quantity of primary copper at the expense of the eutectic, as may be done by lowering the oxide content, the light-grain sections increase in area, changing their contours somewhat; the eutectic filling becomes attenuated until, with an oxygen content of 0.04 per cent. (rather low for commercial tough-pitch copper), a condition similar to that shown at the right in Fig. *b*, Plate 6, is obtained. The connection between the two portions of this figure is easily seen. The filling in of the interstices with pure copper has obscured the principal features of regularity which could be observed at the earlier stage of dendritic growth, but it is highly probable that most of the light sections seen in the right-hand micrograph are parts of the same grain, *i.e.*, possess the same specific orientation.

The eutectic filling is more massive in some places than in others. Where it is concentrated, an intensive rolling operation will result in a relatively broad band of eutectic, *i.e.*, more or less parallel trains of oxide particles in their copper matrix, and, where it is attenuated, narrow bands will result.

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[18] See Mathewson and Caesar, *loc. cit.*

In the present experiments, we started with a planed cake, 2 in. thick, and ended with test strips 0.064 in. thick. Hence, the cake was elongated to some 30 times its original length as a result of the various rolling operations, and equiaxial patches of eutectic are ultimately replaced by bands of varying breadth and thickness depending upon their original dimensions.

A broad band of this sort is seen in Fig. *a*, Plate 4. It is very noticeable that the copper grains in this region are smaller than in adjacent regions free from oxide. Other bands of varying width may be seen in other photomicrographs of the present collection.

In the case of the Copper Range material, as cast (Fig. *a*, Plate 6), three important structural observations may be made: (1) there is no eutectic, (2) a cored structure (shadowy markings) appears, showing variation in internal concentration with respect to arsenic, and (3) the individual oxide particles, which are located with some regularity in the ultimate interstices corresponding to final solidification between and within the grains, are considerably larger than the eutectic particles of oxide in the electrolytic material.[19]

The early annealing of this metal removes all traces of the cored structure and no indication of variable internal composition was observed in the product as received from the rolling mill.

Some fundamental questions of constitution in the system, copper, arsenic, oxygen, have not yet been answered. It is clear that, up to some 3 per cent., the arsenic exists as a solid solution, probably of copper arsenide (Cu_3As) in copper[20] and that arsenic does not deoxidize copper, but exerts a coarsening and scattering effect on the oxide particles,[21] as mentioned above. This latter effect may be due to a greatly increased solubility of cuprous oxide in arsenical copper, whereby the whole amount of oxide (present in tough-pitch metal) remains dissolved until almost all of the copper is frozen, and finally separates out in coarse form from this concentrated solution (as a eutectic containing an unusually high percentage of oxide).

Whatever the constitutional reasons for these facts, it is quite evident that the trains of oxide particles, formed when Copper Range metal is rolled, are always made up of coarse units which do not oppose grain growth as effectively as the highly dispersed form of oxide present in electrolytic copper. Consequently, irregularities of distribution do not, in this case, result in great local variations of grain size, as in the case of electrolytic copper.

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[19] These could not, as a rule, be seen as separate particles under the magnification generally adopted. They are closely sprinkled throughout the dark structure element (eutectic) shown in both parts of Fig. *b*, Plate 6. The conclusions reached above were verified by careful examination of the samples in question at adequate magnifications.

[20] Bengough and Hill: *loc. cit.*

[21] Jolibois and P. Thomas: Du Rôle de l'Arsenic dans les Cuivres industriels, *Revue de Métallurgie*, vol. 10, pp. 1264–1270 (1913).

The principal conclusions drawn from our structural observations are:

(*a*) From the standpoint that uniformity of recrystallized grain size is a mark of quality in an annealed product, metal rolled from the ordinary Copper Range cake is superior to metal rolled from the ordinary electrolytic cake containing about the same percentage of oxide.

(*b*) With regard to the specific effect of included oxide particles on the properties of the finished sheet, bar, or tube, if a small number of coarse particles are less harmful than a large number of fine particles, the Copper Range product is superior to the electrolytic product. Jolibois and Thomas[22] take this view as a matter of course. We are not prepared to make any definite assertion in this respect. It is not improbable that with one form of oxide distribution the metal may be particularly adaptable to one type of service, while the other form may be preferable when the metal must meet requirements of another type.[23]

A fair sample of tube manufactured from Copper Range cake (by hot-rolling, cupping, and closing in) is illustrated by Fig. *c*, Plate 6.

Typical photomicrographs representing the structure of test strips spoiled by annealing in coal gas are shown in Figs. *d* and *f*, Plate 5. The first of these photos shows a sample of electrolytic copper annealed at 800° and photographed at the usual magnification of 68 diameters. This may be compared with Fig. *a*, Plate 4, representing the same material annealed normally at this temperature.

By careful examination of these photomicrographs, it may be seen that many of the boundaries between grains of the "gassed" metal (Fig. *d*, Plate 5) have a dotted appearance, while, in the normally annealed metal (Fig. *a*, Plate 4), these boundaries are close and sharp (the oxide dots are sprinkled at random without regard to the grain boundaries). Observations of this sort are best made at higher magnifications, as illustrated by Figs. *e* and *f*, Plate 5, taken from Copper Range metal annealed at 800° in air and coal gas, respectively. The first figure shows several trains of unaltered oxide particles (magnification, 450), while the last figure shows an abundance of gas cavities located, for the most part, in the grain boundaries. No unaltered oxide (distinguishable by its color, etc.) could be seen in the samples ruined by drastic annealing in coal gas, *i.e.*, in those annealed at 800° and 1,000°, while, in the samples annealed at 600° some unaltered oxide could be seen, along with the gas cavities resulting from reduction of the remainder.

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[22] *Loc. cit.*

[23] Doubtless, in every case, the metal is improved by eliminating oxide. Arsenical copper can be deoxided with phosphorous, the commonest of deoxidizing agents, without excessive loss of arsenic. It is worthy of note that the producers of Copper Range copper profess to be able to bring the metal to pitch with a lower content of oxide than is commonly attained in the case of non-arsenical copper.

Those who have seen Campbell's[24] recent micrograph of over-poled copper may recall its close resemblance to our last micrograph (Fig. *f*, Plate 5) of copper ruined by anneal in coal gas. Deterioration, in both cases, is due to the development of gas within the grains and along the grain boundaries.

Aside from the comparative observations already described, we have made no special study of the effects of annealing under reducing, or other exceptional conditions. It is beyond the scope of this paper to discuss the results obtained by other authors in this particular field.

Our observations indicate that the oxide contained in any of these brands of copper is decomposed by coal gas, even at temperatures of low redness. Presumably, such components of coal gas as can travel freely in copper at red heat (certainly hydrogen, probably carbon monoxide, and perhaps certain hydrocarbons) effect an ordinary reduction of the oxide, with formation of bulky gaseous products which seem to accumulate in the grain boundaries,[25] but also occur abundantly within the grain substance.

The character of the fractures indicates that the tensile break occurs very largely at the grain boundaries through the gas cavities, whose lining seems to be devoid of luster.

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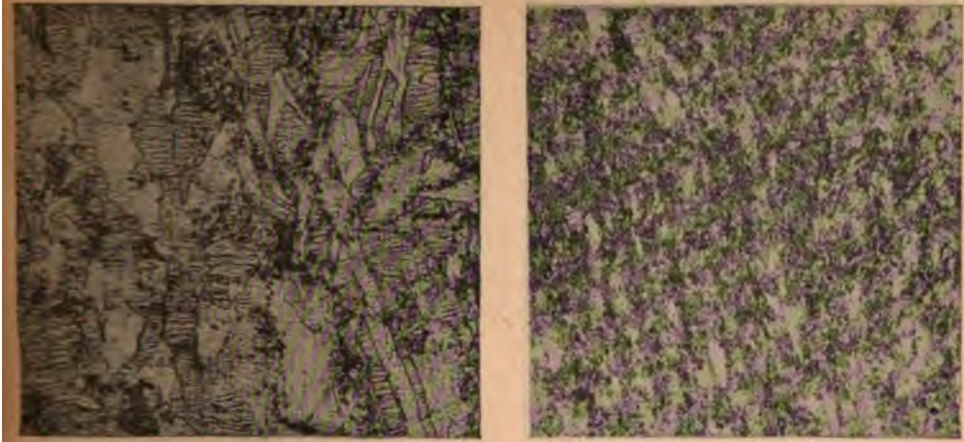
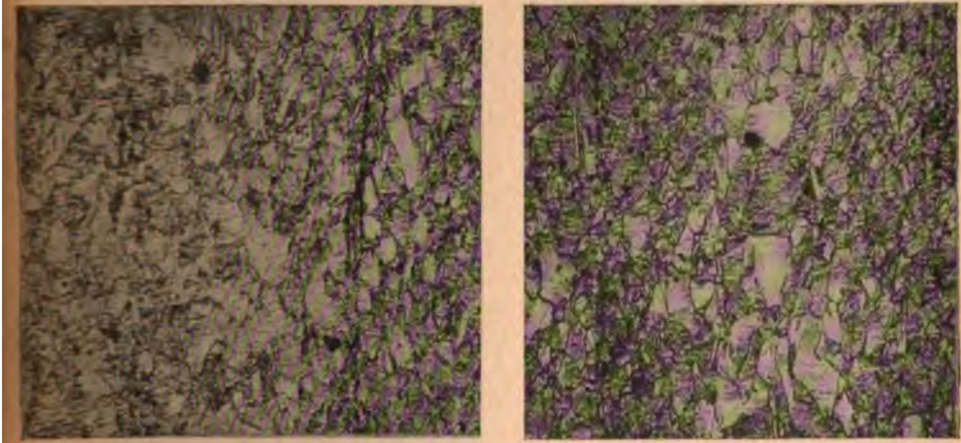
[24] *Loc. cit.*

[25] G. D. Bengough and D. Hanson's tensile tests (*Journal Institute of Metals*, vol. 12, pp. 56–88, 1914) have shown that these boundaries are favored regions of weakness at high temperatures (the grains pull apart at the boundaries), which indicates that gas evolved from within would find its path of least resistance along the boundaries.



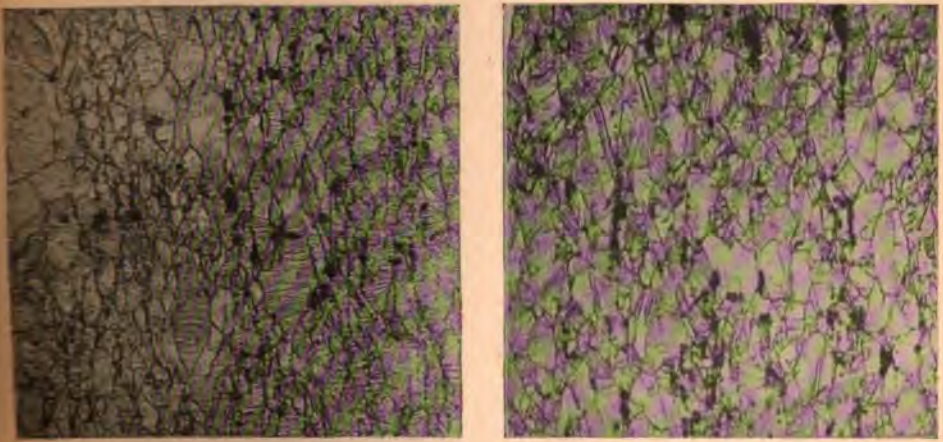
a.—Electrolytic Copper, 200° C. × 68.

d.—Electrolytic Copper, 300° C. × 77.



b.—Mohawk Copper, 200° C. × 68.

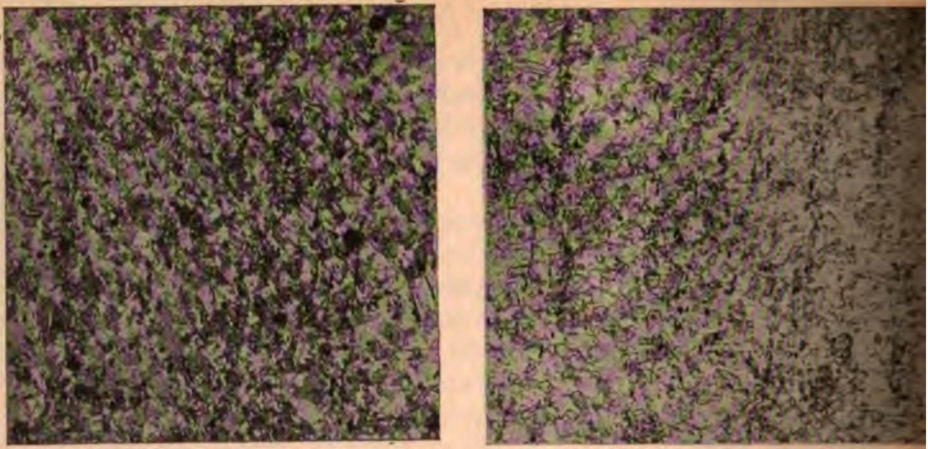
e.—Mohawk Copper, 300° C. × 77.



c.—Copper Range Copper, 200° C. × 68.

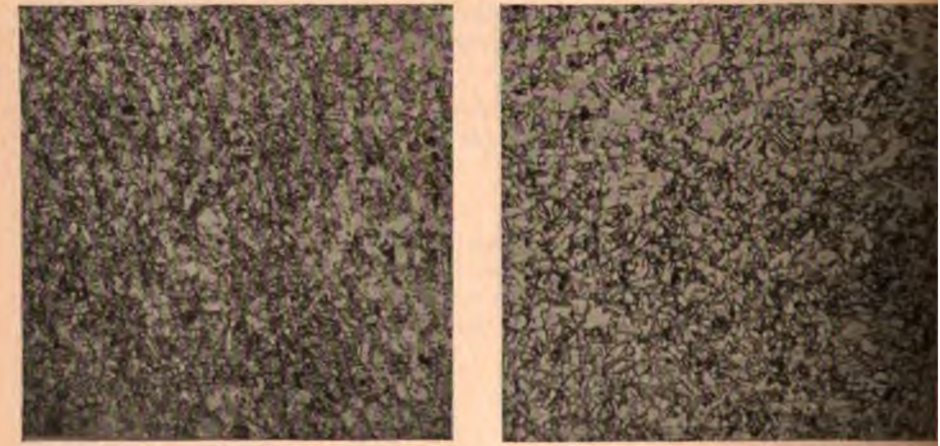
f.—Copper Range Copper, 200° C. × 77.

PLATE 1.—COPPER ANNEALED AT 200 AND 300° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



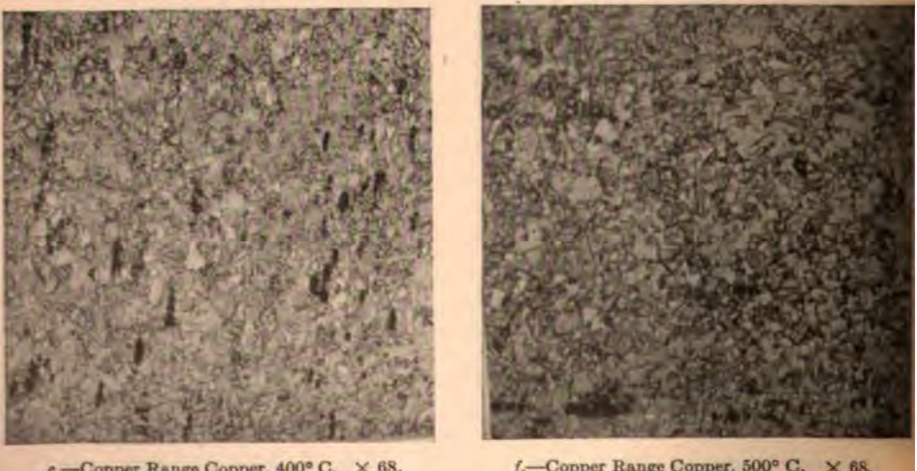
a.—Electrolytic Copper, 400° C. × 68.

d.—Electrolytic Copper, 500° C. × 68.



b.—Mohawk Copper, 400° C. × 68.

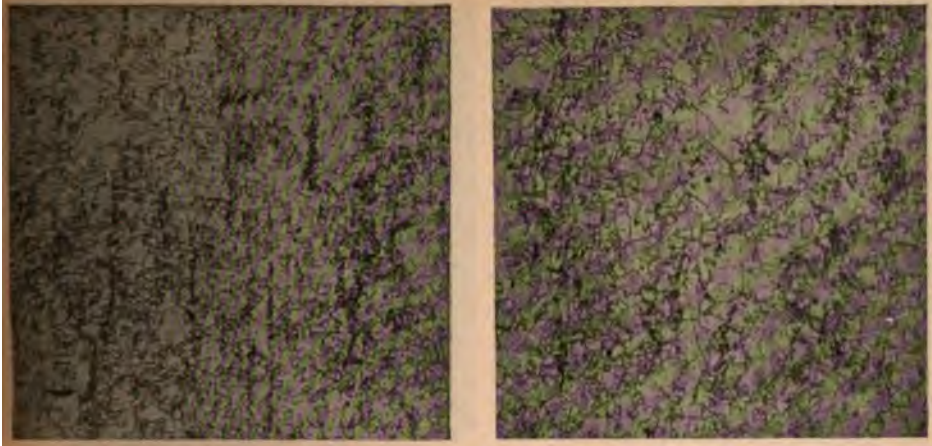
e.—Mohawk Copper, 500° C. × 68.



c.—Copper Range Copper, 400° C. × 68.

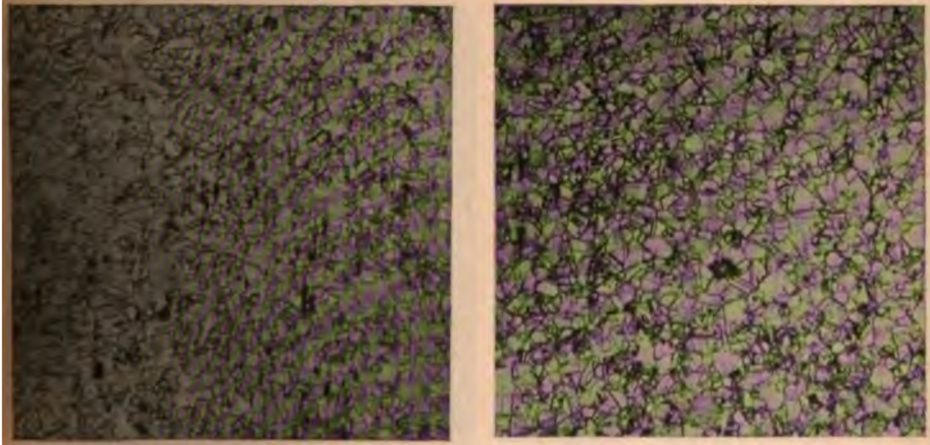
f.—Copper Range Copper, 500° C. × 68.

PLATE 2.—COPPER ANNEALED AT 400 AND 500° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



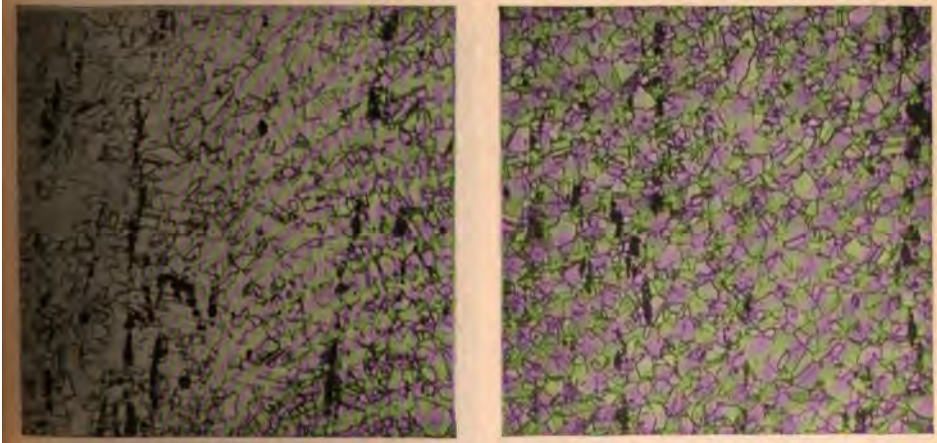
a.—Electrolytic Copper, 600° C. × 68.

d.—Electrolytic Copper, 700° C. × 68.



b.—Mohawk Copper, 600° C. × 68.

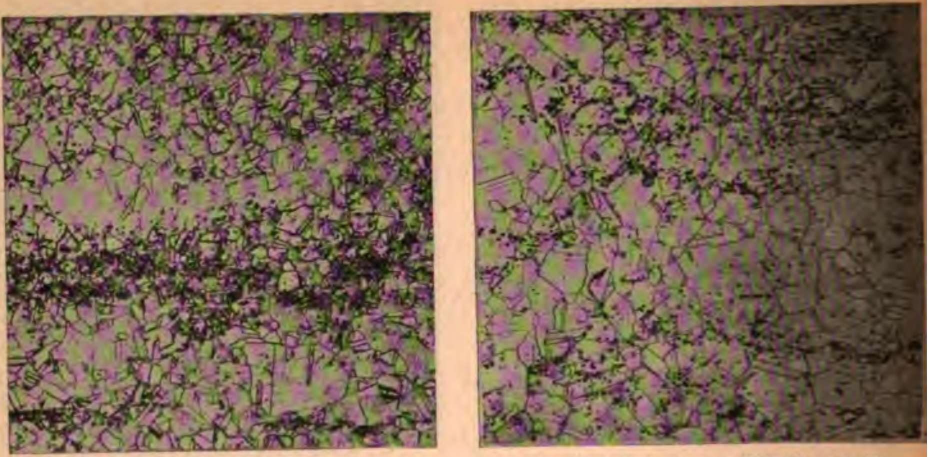
e.—Mohawk Copper, 700° C. × 68.



c.—Copper Range Copper, 600° C. × 68.

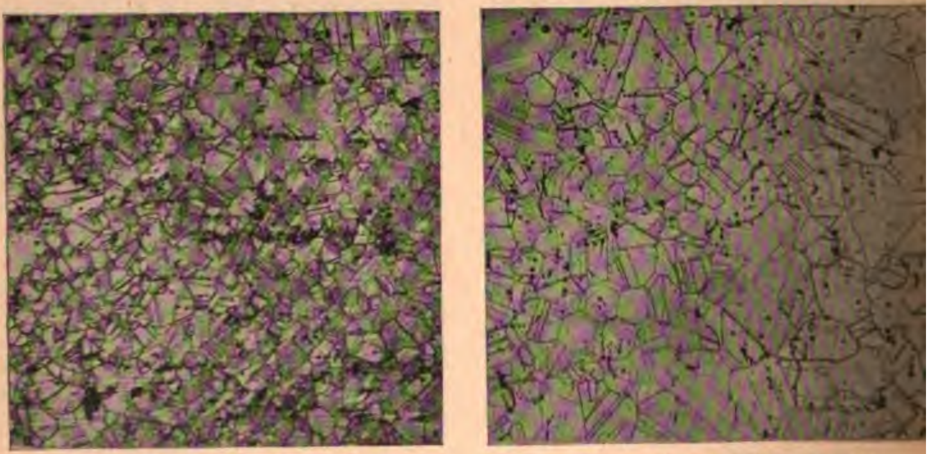
f.—Copper Range Copper, 700° C. × 68.

Plate 3.—Copper Annealed at 600 and 700° C. Samples Etched with Ammonia and Hydrogen Peroxide.



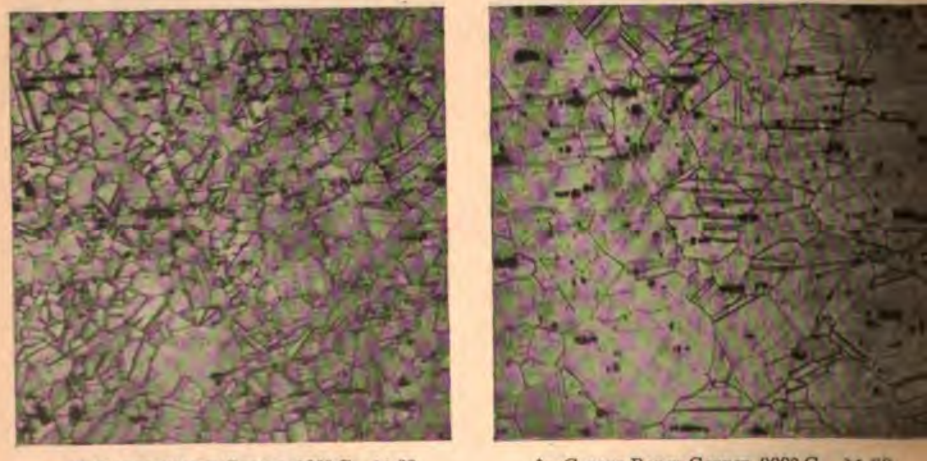
a.—Electrolytic Copper, 800° C. × 68.

d.—Electrolytic Copper, 900° C. × 68.



b.—Mohawk Copper, 800° C. × 68.

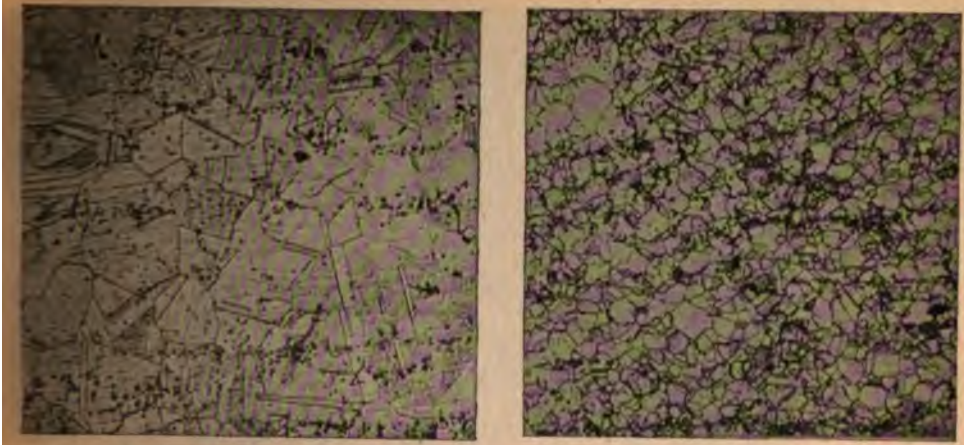
e.—Mohawk Copper, 900° C. × 68.



c.—Copper Range Copper, 800° C. × 68.

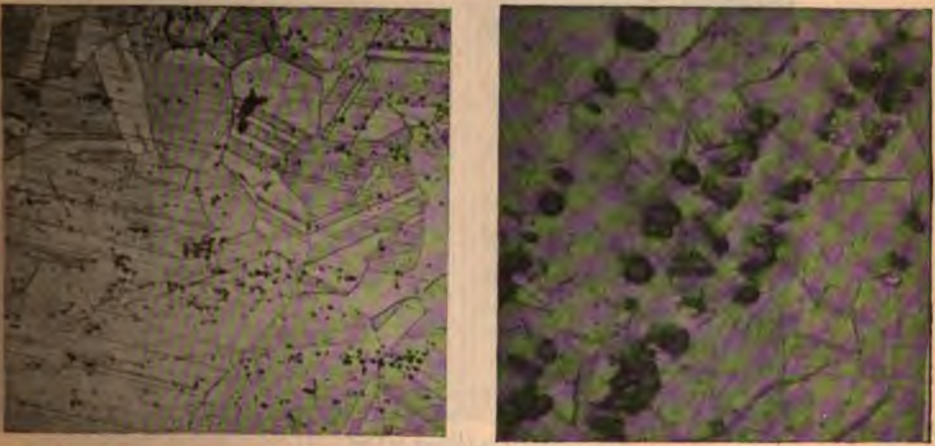
f.—Copper Range Copper, 900° C. × 68.

PLATE 4.—COPPER ANNEALED AT 800 AND 900° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



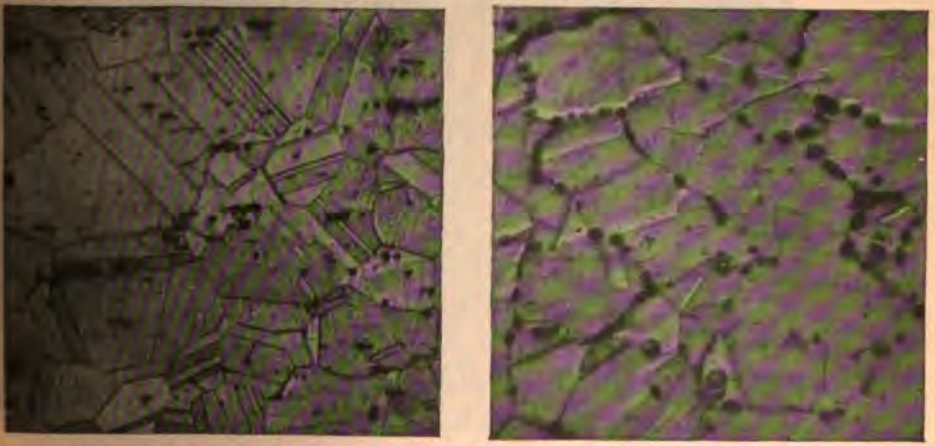
a.—Electrolytic Copper, 1000° C. ×68.

d.—Electrolytic Copper, Annealed at 800° C. in Coal Gas. × 68.



b.—Mohawk Copper, 1000° C. × 68.

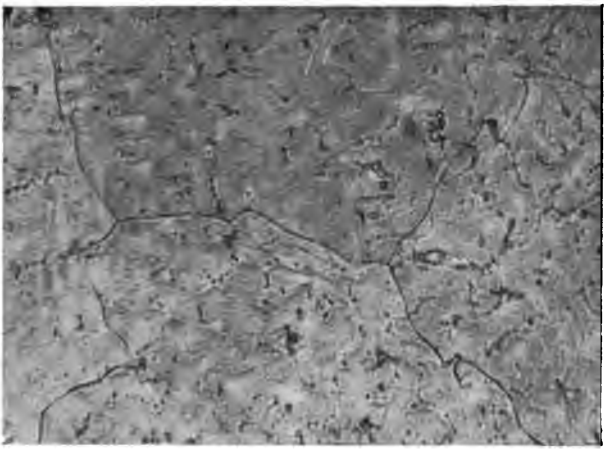
e.—Copper Range Copper, Annealed at 800° C. in Air. × 450.



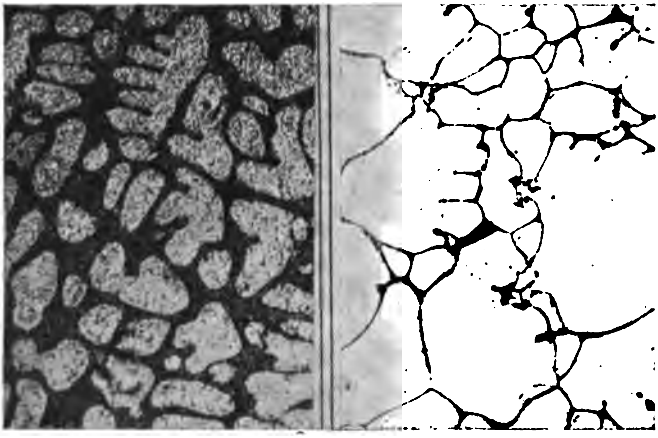
c.—Copper Range Copper, 1000° C. × 68.

f.—Copper Range Copper, Annealed at 800° C. in Coal Gas. × 450.

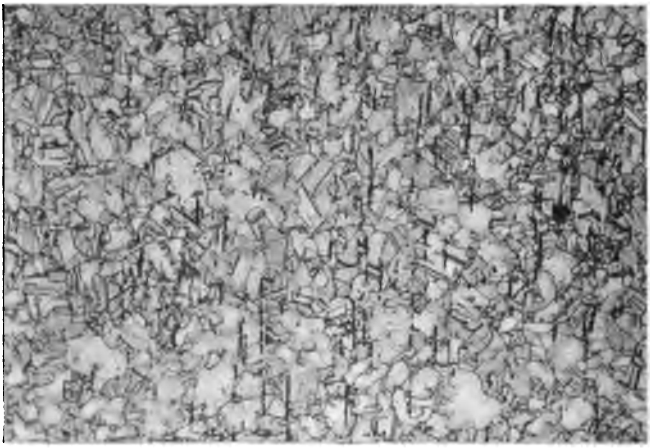
Plate 5.—Copper Annealed at 1000° C in Air, and Samples Annealed at 800° C. in Air and Coal Gas. Samples Etched with Ammonia and Hydrogen Peroxide.



a.—Copper Range Copper as Cast. 0.055 Per Cent. Oxygen.



b.—Electrolytic Copper as Cast.
0.14 Per Cent. Oxygen. 0.04 Per Cent. Oxygen.



c.—Annealed Tube from Copper Range Copper.

Plate 6.—Samples Etched with Ammonia and Hydrogen Peroxide. × 69.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Tungsten-Molybdenum Equilibrium Diagram and System of Crystallization

BY ZAY JEFFRIES,* B.S., MET. E., CLEVELAND, OHIO

(Arizona Meeting, September, 1916)

In this paper, it is proposed to outline a method for the determination of melting points of those metals and alloys having high fusion temperatures. The application of the method as used to determine the melting points of alloys of tungsten and molybdenum will be given. It will be seen from these results that tungsten and molybdenum form a completely isomorphous series. This is also verified by the accompanying photomicrographs.

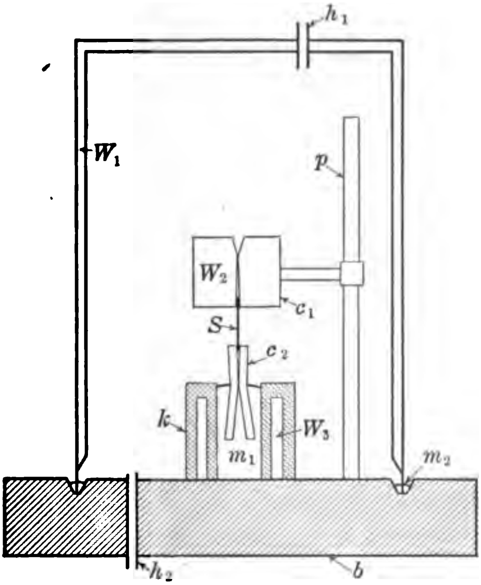


Fig. 1.—Partial Section of Apparatus Used to Determine Melting Points of Tungsten and Molybdenum Alloys.

Tungsten and molybdenum have been found to crystallize in the same system, namely, isometric, and the crystal units are cubes. These determinations were made by examining etching pits in the pure metals.

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* Case School of Applied Science.

Description of Apparatus

Fig. 1 shows a partial section of the apparatus used for determining the melting points of the alloys. W_1 is a water-jacketed metal housing with a mercury seal $\bar{m}_2$ at the bottom. $\bar{h}_1$ is a hydrogen inlet and $\bar{h}_2$ a hydrogen outlet. $\bar{c}_1$ is the top electrode which is water-cooled, as indicated by W_2. This electrode is supported by the post $\bar{p}$, to which it is fastened by means of an adjustable sleeve. $\bar{c}_2$ is the bottom electrode which floats in a bath of mercury $\bar{m}_1$. This mercury is contained in the casting $\bar{k}$, which is water-cooled, as shown by W_3. The whole apparatus is mounted on the base $\bar{b}$ made of electrical insulating material. The electrodes are suitably connected to a current regulator, and the voltmeter and ammeter are properly inserted in the circuit.

Method of Making Determinations

Wires (S), 2¾ in. between electrode contacts and 0.030 in. in diameter, were inserted in the electrodes and hydrogen was allowed to flow through the inclosed chamber until the air was expelled. Electric current was then passed through the wire, starting with low amperage and gradually increasing until fusion occurred. The time of fusion is easily determined because the circuit is broken and the ammeter ceases to register. The readings of the voltmeter and ammeter are recorded with each increase in electric current so the fusion wattage can be accurately calculated.

Besides using samples of the same size, other conditions must be maintained constant; for example, the flow of hydrogen through the inclosed chamber, the flow of water through each of the water jackets, and the temperature of the water at the intake. The electrical resistance of the electrodes and their connections is so small that it can be neglected in the calculation for drop of potential when compared to the drop of potential between the two electrodes.

Calibration of the Apparatus using Samples of Known Melting Points

For calibrating the apparatus wires of pure platinum, pure molybdenum and pure tungsten were used. These wires were all of the same size and same length. The number of watts to fuse each is shown in Table 1.

TABLE 1

Material	Fusion Wattage
Platinum	144
Molybdenum	400
Tungsten	1,800

By considering the fusion wattage of tungsten as 100 per cent., the fusion wattage of molybdenum is 22.2 per cent. and of platinum, 8 per cent.

The curve in Fig. 2 is obtained by plotting the following temperatures against the corresponding fusion wattage percentages:

TABLE 2

Room Temperature, Degrees Centigrade	Per Cent. of Tungsten Fusion Wattage
20	0.0
1,755	8.0
2,500	22.2
3,300	100.0

The temperature 3,300° C. is recommended as the most probable figure for the melting point of tungsten by Dr. Langmuir,[1] and also by the Nela Laboratories.[2]

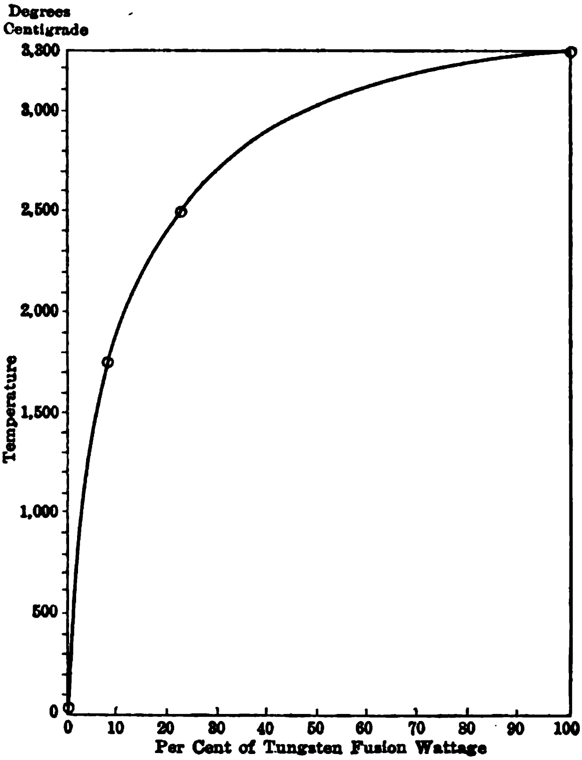

FIG. 2.—PER CENT. OF TUNGSTEN FUSION WATTAGE AT DIFFERENT TEMPERATURES.

If a sample of unknown melting point, having the same diameter and same length as the standardizing samples, be fused in the apparatus, its fusion temperature can be interpolated from its fusion wattage, from the curve shown in Fig. 2. A discussion will be found below outlining the possible sources of error in the determination of melting points by this method.

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[1] I. Langmuir: Melting Point of Tungsten, *Physical Review*, Series 2, pp. 138–157 (August, 1915).

[2] A. G. Worthing: The True Temperature Scale for Tungsten and its Emissive Powers at Incandescent Temperatures, *Journal of the Franklin Institute*, vol. 181, No. 3, pp. 417–418 (March, 1916).

Results Obtained with Tungsten-Molybdenum Alloys

The results are shown in Table 3, and the points on the equilibrium diagram are shown in Fig. 3.

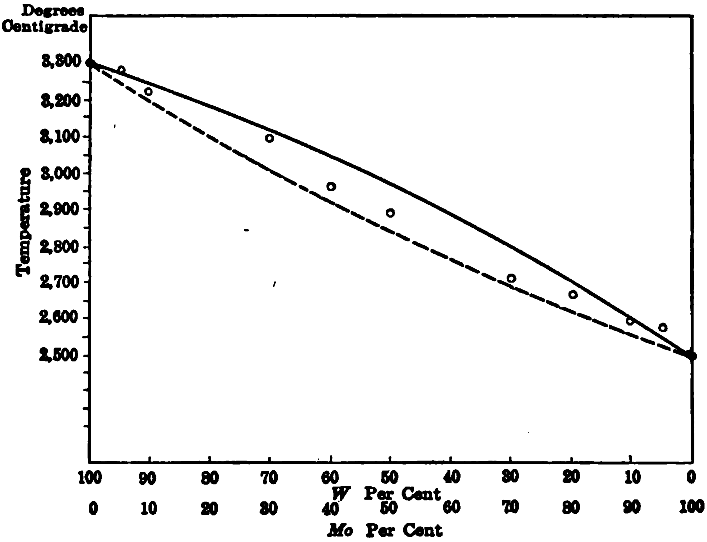

FIG. 3.—EQUILIBRIUM DIAGRAM OF TUNGSTEN AND MOLYBDENUM.

TABLE 3.—*Fusion Results Obtained with Tungsten-Molybdenum Alloys*

Molybdenum, Per Cent.	Tungsten, Per Cent.	Watts to Fuse	Per Cent. of Tungsten Fusion Wattage	Fusion Temperature, Degrees Centigrade
100	0	400	22.2	2,500
99	1	410	22.8	2,510
95	5	445	24.7	2,580
90	10	452	25.2	2,600
80	20	509	28.25	2,670
70	30	541	30.1	2,710
50	50	710	39.4	2,890
40	60	797	44.2	2,960
30	70	1,003	57.2	3,100
10	90	1,341	74.5	3,220
5	95	1,650	91.7	3,280
1	99	1,768	98.2	3,295
0	100	1,800	100.0	3,000

The solidus and liquidus curves are drawn tentatively, it being, of course, impossible to determine either of these curves by this method.

The point of breakdown of the alloys under the action of the electric current probably lies between the solidus and the liquidus. Most of the points shown in the diagram in Fig. 3 lie between the proposed solidus and liquidus curves. The fact that two points lie outside of the area included between these two curves may be explained by saying that the accuracy of any of the determinations may be subject to an error of from 30 to 50° C.

As an example to show how the temperature is obtained from the fusion wattage, let us consider the alloy containing 50 per cent. tungsten and 50 per cent. molybdenum. The fusion wattage of this alloy was 710, and the fusion wattage of tungsten was 1,800.

$$\frac{710}{1,800} \times 100 = 39.4$$

This alloy fuses, therefore, at 39.4 per cent. of the fusion wattage of tungsten. Referring to Fig. 2, it will be seen that 39.4 per cent. of the fusion wattage of tungsten corresponds to a temperature of about 2,890° C.

The equilibrium diagram suggests that tungsten and molybdenum form a series of alloys which are completely soluble in each other, both in the liquid and solid states, and that the melting point of any alloy of the series will lie between the melting points of the end members. The photomicrographs confirm this indication.

Mennicke[3] observes that two compounds between tungsten and molybdenum occur, namely, W_2Mo_3 and WMo. The former would contain 79.3 per cent. tungsten and the latter 65.7 per cent. Samples closely approximating these analyses, as well as those containing more and less tungsten, have solid-solution structures. All of the evidence of the melting points indicates that tungsten and molybdenum form no intermetallic compounds.

Microscopic Examination of Tungsten and Molybdenum Alloys

Fig. 4 is a photomicrograph of nearly pure tungsten at a magnification of 320 diameters. The impurity in this sample is non-metallic. A few globules of this can be seen in the micrograph. This sample contains by analysis about 99.8 tungsten.

Figs. 5 to 7 inclusive are photomicrographs at 320 diameters of the various alloys of tungsten and molybdenum, as indicated in the accompanying descriptions. It can be readily seen that all of these alloys are solid solutions.

Judging from the micrograph (not here reproduced) alone, of an alloy of 20 per cent. tungsten and 80 molybdenum, the black areas might be considered to be due to a second phase; in fact, these regions represent areas rich in tungsten. Under high power (1,200 diameters) the separation of the two phases could readily be detected. That they were not in equilibrium was also demonstrated by an additional heating which completely homogenized the alloy. In this connection, it would probably be advisable to mention the manner in which these alloys were made.[4]

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[3] H. Mennicke: *Die Métallurgie des Wolframs*, p. 262. Berlin, 1911.

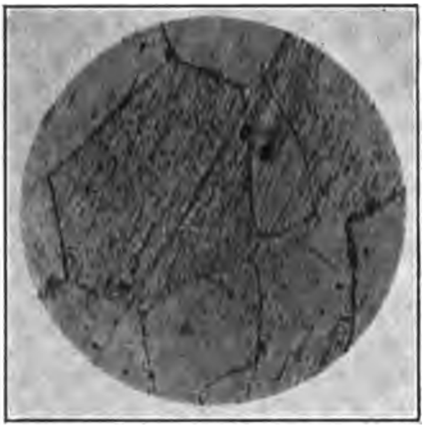

FIG. 4.—PURE TUNGSTEN. × 320.

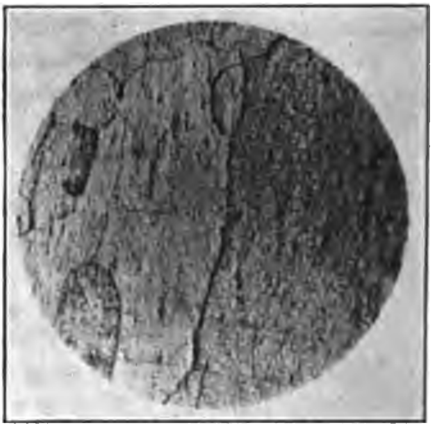

FIG. 5.—TUNGSTEN, 90 PER CENT.; MOLYBDENUM, 10 PER CENT. × 320.

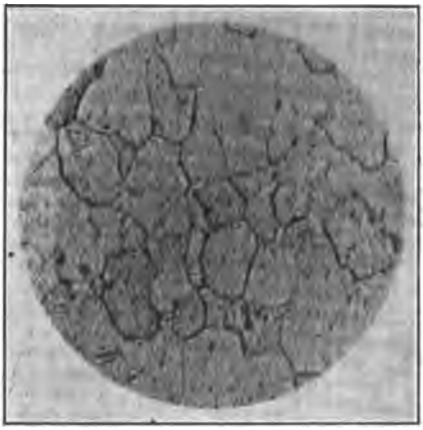

FIG. 6.—TUNGSTEN, 75 PER CENT.; MOLYBDENUM, 25 PER CENT. × 320.

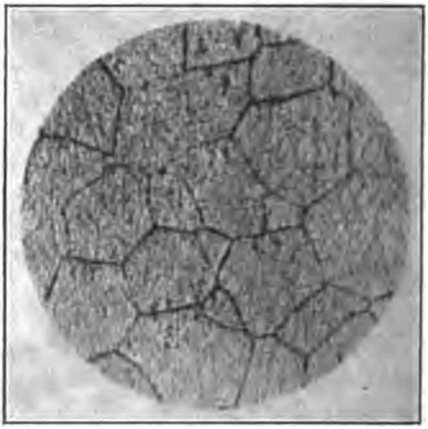

FIG. 7.—TUNGSTEN, 1 PER CENT.; MOLYBDENUM, 99 PER CENT. × 320.

The oxides of both tungsten and molybdenum were obtained in a very pure powdered state, and reduced to powdered metal by hydrogen.

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[4] The method of making and swaging these alloys was practically the same as described by R. W. Moore, *Metallurgical and Chemical Engineering*, vol. 12, p. 186 (March, 1914).

The alloys were made up by thoroughly mixing the tungsten and molybdenum powders in the proper amounts. The mixed powders were then pressed into briquets which were sintered at about 1,300° C. in an atmosphere of hydrogen and were then heated by electric current to about 100° C. below the fusion points (which were previously determined with a separate series of briquets of these alloys) for a period of 12 min. The alloys, after receiving the above treatment, were completely crystalline. They may be ground by an emery wheel to a certain diameter, and the fusion points determined on these samples. They may also be heated and rolled or swaged to any desired diameter. The alloys used in these experiments were swaged to 0.030 in. diameter, the original cross-section of the briquets being about 1/16 sq. in.

The micrographs, with the exception of Fig. 6 (which is a section of 0.080 in. wire), represent longitudinal sections of these 0.030 in. diameter wires after fusion. The portions shown in these micrographs represent sections which have not been fused but which have been heated to temperatures very near their melting points. It will be evident that fusion of the wire will take place at a point approximately equidistant between the two electrodes and that adjacent portions of the wires would be heated to very near the melting point. In these sections, therefore, the solution of the tungsten and molybdenum must take place while both metals are in the solid state, or at least while the tungsten-rich portion is in the solid state. In case solution and diffusion have not been complete, two phases, one rich in tungsten, the other rich in molybdenum, will be present. In some of the briquets, the solution was not complete during the first heating, but was complete after the second heating. All evidences of

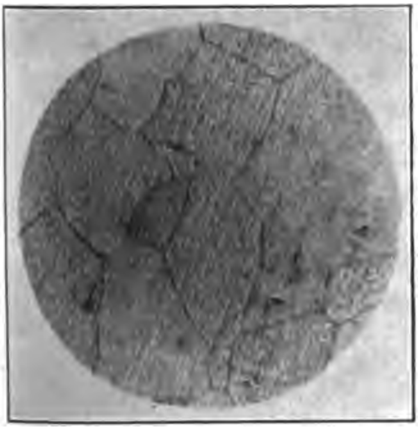

FIG. 8.—PURE MOLYBDENUM. × 320.

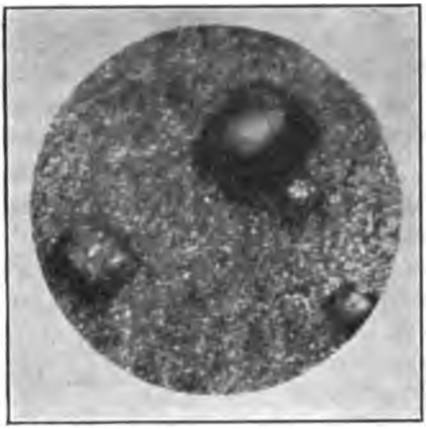

FIG. 9.—ETCHING PITS IN TUNGSTEN. × 775.

the existence of two phases were removed when sufficient time was given the samples at temperatures near their melting points.

These samples were also examined after fusion in the parts which had been molten. There was no evidence of a second phase in any of these samples; that is, they were all true solid solutions.

Fig. 8 represents a micrograph of pure molybdenum. By analysis, it contains about 99.9 per cent. molybdenum. Its purity is also suggested by the micrograph.

These alloys are all readily etched by boiling hydrogen peroxide. Etching pits in the various alloys were produced by a comparatively long attack—say, 3 or 4 min. in boiling hydrogen peroxide. All of the evidence from the etching pits points to the conclusion that the crystal units are cubes. The alloys of tungsten and molybdenum, when made up according to the above description, seem to form etching pits very much more readily than either of the pure metals. This might be attributed to the existence of small particles of either the tungsten-rich or the molybdenum-rich portions, which, by the difference in composition, would facilitate solution at that point, thus easily forming the etching pits.

It is of interest in this connection to note that the average grain as seen in the micrographs represents about 1,000 of the original particles of the powdered metals. This free grain growth is positive evidence of free solution of the metals in each other. Had the metals remained as separate phases, each would have impeded the coalescence of the grains of the other.

Probable Accuracy of Results

The quantity of heat represented by the fusion wattage is independent of the quantity of heat actually necessary to raise a mass of metal, similar to that used in these samples, up to its melting point. It depends on the ability of the sample under investigation to dissipate heat in the apparatus. The thermal balance may be expressed by the equation:

Heat added electrically =
heat dissipated by radiation, convection and conduction.

The conditions obtaining within the apparatus are such that if the fusion of tungsten is represented by 1,800 watts, 1,750 watts could be dissipated for a long period of time without fusion of the tungsten.

The probable errors in the determinations are outlined below:

Errors due to:

1. Differences in emissivity of the various alloys under investigation.
2. Differences in their specific heats, heat capacities and heat conductivities.
3. The selective volatilization of molybdenum.

4. Variations in the flow of water through the water jackets, of the temperature of the water at the intake, and of the flow of hydrogen through the inclosed chamber.

5. Slight variations in the diameter of the wires.

6. Possible changes of melting points due to chemical or physical combination of alloys with hydrogen.

7. The personal equation in drawing the per cent. of tungsten fusion wattage-temperature curve.

8. The readings of the electrical instruments.

The differences in emissivity of the various alloys will make but a slight difference in the fusion wattage, for the reason that only about one-twentieth of the total heat dissipated is lost by radiation, and the electrodes combined.

The errors due to No. 2 will be very slight, for the same reason.

Langmuir and McKay[5] report that at 2,800° K., the heat loss from a tungsten filament in hydrogen at 750 mm. pressure, is 10½ times greater than that due to radiation; and at 3,500° K., the heat loss due to hydrogen is 11½ times that due to radiation. The conditions obtaining in the apparatus used by the writer were such that the losses due to the hydrogen are considerably more than is reported by these authors.

Dr. Irving Langmuir[6] suggests errors due to the selective volatilization of molybdenum.[7] The apparent high melting points of the alloys containing 5 per cent. tungsten, 95 per cent. molybdenum, and 5 per cent. molybdenum, 95 per cent. tungsten, may be partially due to this selective volatilization. The enrichment of the alloys in tungsten, due to this cause, could not have been more than 3 per cent. in any case, as was indicated by the diameter measurements of the wires before and after fusion.

The errors due to No. 4 are probably greater than those due to all other variations combined. Errors due to these causes, however, are not necessary if the proper precautions are taken. These precautions would consist of supplying the water for the water jackets from a constant-level and constant-temperature tank. The same conditions should be fulfilled with the hydrogen.

To find out the approximate magnitude of these errors, samples of the same wires were fused on different days, so that the valve-settings for jacket water and hydrogen had to be re-made and re-adjusted.

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[5] I. Langmuir and G. M. J. Mackay: The Dissociation of Hydrogen into Atoms, Pt. I, *Journal of the American Chemical Society*, vol. 36, pp. 1708–1722 (1914).

[6] Private communication, March 29, 1916.

[7] I. Langmuir: The Vapor Pressure of Metallic Tungsten, *Physical Review*, Series II, vol. 2, No. 5, pp. 329–342 (November, 1913).

I. Langmuir and G. M. J. Mackay: The Vapor Pressure of the Metals Platinum and Molybdenum, *Physical Review*, Series II, vol. 4, No. 4, pp. 377–386 (October, 1914).

Several determinations showed a maximum variation in the melting point of molybdenum, as interpolated from the curve in Fig. 2, of about 40° C., and a maximum variation in the melting point of tungsten of about 10° C. The results on tungsten seem very favorable when it is considered that some methods for determining its melting point may vary 150° C.

Allowing for other errors, encountered in the determinations, it is reasonably accurate to assume that the maximum errors toward the molybdenum side of the diagram, are about 50° C., and toward the tungsten side of the diagram, 30°C. Allowing for errors of this magnitude, the points in Fig. 3 would fit nicely between the solidus and liquidus curves.

Errors due to slight variations in the diameters of the wires will be very small, owing to the fact that the ability of small wires to dissipate heat in a gas depends largely upon the existence of a relatively thick film of gas surrounding the wire. The thickness of this gas film is dependent largely upon the pressure of the gas, and not upon the diameter of the wire. The fact that these errors were slight was confirmed by an experiment, and was also predicted by Dr. Langmuir[8] from his extensive work along this line.

The suggestion was made by Dr. G. K. Burgess[9] that hydrogen might, by reason of either chemical or physical combination with the alloys, change their melting points. The writer has no knowledge concerning this. It might be well to add, however, that the handling of this type of alloys at high temperatures is done almost entirely in hydrogen.

To ascertain the approximate errors due to the drawing of the curve in Fig. 2, it was plotted independently on two sheets of paper. The maximum difference between the two curves was about 10° C. at about 60 per cent. of the tungsten fusion wattage. To facilitate the interpolation of temperatures from fusion wattages, this curve was drawn on a large piece of cross-section paper, so that the reading errors would be reduced to a minimum.

The wattage, as determined by volts and amperes, was, of course, quite accurate, so errors from this source were negligible.

Concerning the probable accuracy of results obtained by this method, Dr. Irving Langmuir writes: "I think the method you are using for estimating the temperature of filaments should give reasonably accurate results." He points out, among other things, the desirability of making chemical analyses *after* heating, to determine the actual composition of the alloy at the time of fusion. Dr. G. K. Burgess writes: "This method seems to me susceptible of very considerable accuracy."

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[8] I. Langmuir: The Dissociation of Hydrogen into Atoms, Pt. II., *Journal of the American Chemical Society*, vol. 37, pp. 417–458 (March, 1915).

[9] Private communication, April 6, 1916.

The author is convinced that the method is susceptible of greater accuracy than was obtained; that, with careful attention given to the construction and operation of the apparatus, melting points can be determined within 10° C. The method promises to offer solutions for several equilibrium diagrams of the higher melting-point metals.

The Crystal System of Tungsten and Molybdenum

Fig. 9 shows etching pits in tungsten, magnified 775 diameters. As can readily be seen from the micrograph, these etching pits represent sections of cubes cut by a plane parallel to a face.

Fig. 10 shows triangular etching pits which are formed by a plane cutting three faces of a cube.

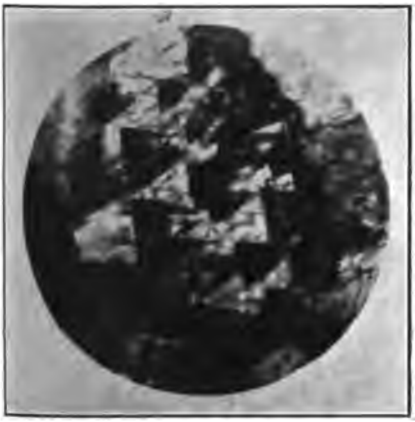

Fig. 10.—Etching Pits in Tungsten. × 775.

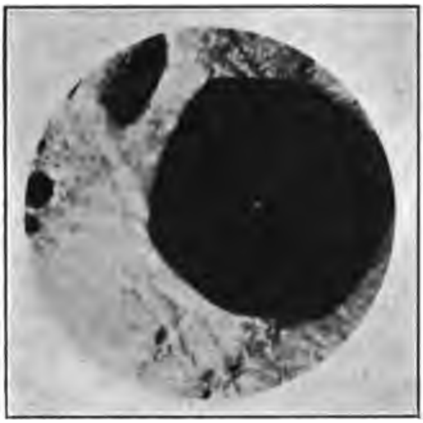

Fig. 11.—Etching Pits in Molybdenum. × 775.

Fig. 11 represents the intersection of the six faces of a cube by a plane. By changing the direction of the illumination, the writer was able to examine the bottom of this etching pit and could easily see that it was a section of a cube.

It was noticed in some of the micrographs that the etching pits instead of having straight line sides had curved sides. Goldschmidt[10] thinks that these curved lines are due to the convection currents set up in the etching solution by the unequal rate of attack at various points. The solution will be used up most rapidly at places where the greatest amount of surface is exposed. This will cause a flow of fresh solvent by diffusion. The corners of the etching pits will offer greater resistance to the flow of convection currents than the adjacent portions, and hence will not receive so much fresh solution. The lines bounding the polygons are thus dissolved at unequal rates and become curved instead of straight.

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[10] V. Goldschmidt: From the Borderland between Crystallography and Chemistry, *Bulletin University of Wisconsin*, Science Series, vol. 3, No. 2, pp. 21–38 (March, 1908).

A great many etching pits in all conceivable positions were examined microscopically. From these examinations, the author has very little hesitancy in saying that both tungsten and molybdenum crystallize in the isometric system and the form of their crystal units is the cube.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Zircon-Bearing Pegmatites in Virginia

BY THOMAS L. WATSON,* PH. D., CHARLOTTESVILLE, VA.

(Arizona Meeting, September, 1916)

Introduction

THE occurrence of zircon in pegmatites of acidic composition is recorded by many observers both in this country and abroad, and they form one of the most important geologic modes of occurrence of the mineral.[1] As is well known, commercially the most important American locality for zircon is near Zirconia in Henderson County, North Carolina, where many tons of the mineral have been obtained from a kaolinized pegmatite dike 100 ft. wide and traced for 1½ miles along the direction of strike N. 50° E.

Zircon may be developed in pegmatites as inclusions in the principal rock-forming minerals, chiefly quartz and feldspar, and as a separate megascopic constituent in the form of grains and crystals not exceeding, as a rule, 2 in. in size, and usually smaller. In the Henderson County, North Carolina, pegmatite, zircon occurs in prismatic crystals with pyramidal terminations measuring up to 30 mm. in diameter, associated chiefly with the feldspar. In some of the apatite veins of Canada, which are closely allied to pegmatite dikes, zircon is reported in crystals upward of 6 in. in length and 2 in. and more in thickness.

Zircon in large masses occurs in Virginia in the well-known pegmatites near Amelia in Amelia County, and near Gouldin in Hanover County. Both localities are in the middle eastern portion of the Piedmont Plateau province, and are separated by a distance of about 40 miles in a northeast-southwest direction. The rocks of both areas, including the pegmatite bodies, are in an advanced stage of decay, and the hard and moderately fresh rocks are concealed beneath a cover of variable thickness of rock decay derived by the normal processes of weathering. Exposures, therefore, of even moderately fresh rock in the two areas are rare except along stream courses.

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* Professor of Geology, University of Virginia, and State Geologist of Virginia.

[1] For a summary of the properties, occurrence, and uses of zircon, see paper by T. L. Watson and F. L. Hess on Zirconiferous Sandstone near Ashland, Virginia, in *Bulletin of Philosophical Society of the University of Virginia*, Scientific Section, vol. 1, No. 11, pp. 267–292. (1912.)

Pegmatite dikes occur in most of the Virginia Piedmont counties, and the detailed study of them now in progress by the State Geological Survey will doubtless add to those of Amelia and Hanover Counties that are zircon-bearing. The particularly interesting features of the zircon noted in the pegmatites of the two Virginia localities are (1) its occurrence in massive forms of unusual size, and (2) its association in the two localities with an entirely different group of rarer minerals, although the pegmatites of each area are of granitic composition. Zircon and apatite are the only rare minerals that have been found alike in the two areas, and, so far as the pegmatite bodies have been worked, zircon is more abundant in the pegmatites of the Hanover County area than in those of Amelia County.

Amelia County Area

The pegmatite bodies occurring near Amelia Courthouse in Amelia County have long been known for the variety of rare minerals found in them, many of which were of unusual size. The dikes have been worked from time to time for a long period of years as a source of commercial mica and feldspar, and to a less extent of minerals for the gem trade.

The country rock is a thinly foliated, moderately dark-colored, fine-grained, biotite gneiss or schist, containing more or less muscovite. Where measured, the foliation strikes N. 25° to 30° E. and dips 40° to 50° N. W. Diabase dikes of Mesozoic age intrude the rocks in places. The pegmatite bodies are dike-like in form and nearly vertical, with the direction of trend doubtful. They cut across the foliation of the schists and the large ones will measure more than 50 ft. across. They are cut by joints, but there is no evidence of schistose structure developed from metamorphism.

The pegmatites are of granitic (acidic) composition, containing feldspar, including the potash varieties, orthoclase and green microcline, and the soda variety, albite, with quartz and muscovite, and a large number of rarer minerals. The principal rock-forming minerals are not uniformly distributed through all parts of the pegmatites, but their distribution is very irregular, first one and then another of these minerals predominating in different parts. The albite, occurring in splendid crystallizations as reticulated platy forms of bluish white to white color and frequently transparent, is of a high degree of purity as indicated in the two following analyses.

The texture of the pegmatites is granular consertal rather than graphic. Miarolitic cavities have been observed in some of the openings made in the pegmatites. One of these was of large size, the walls of which were lined with crystals of smoky quartz and pure white crystals of albite, some as transparent as glass.[2]

Analyses of Albite from Amelia County, Virginia

	I	II
SiO_2	68.44	68.22
Al_2O_3	19.35	19.06
Fe_2O_3	.....	0.15
CaO	.....	0.40
Na_2O	11.67	11.47
K_2O	0.43	0.20
H_2O	.....	0.69
	99.89	100.19
Specific gravity	2.605	

I. R. N. Musgrave, *Chemical News*, vol. 46, p. 204 (Nov. 3, 1882).
II. E. T. Allen, *Bulletin No.* 591 *of U. S. Geological Survey*, p. 300 (1915).

The rarer minerals include representatives of five distinct chemical groups: (1) *Haloids,* including fluorite; (2) *silicates,* including garnet (spessartite), black tourmaline, beryl, helvite, allanite, and zircon; (3) *niobates,* including columbite; (4) *tantalates,* including microlite; and (5) *phosphates,* including apatite and monazite. With the exception of fluorite, tourmaline, and zircon, each of the minerals has been analyzed with the results shown below. Some of these minerals have been found only occasionally in the Amelia pegmatites and are very rare. Many of them attained unusual size, such as crystals of beryl 3 to 4 ft. long and 18 in. thick, columbite in crystalline masses weighing 6 to 8 lb., allanite crystals more than 15 in. long, microlite in masses up to 8 lb. in weight, and monazite in masses larger than those of microlite.[3] Zircon has been noted in small crystals and in masses weighing several pounds. Stibnite and galena have been reported, but they are extremely rare and have not been seen by the writer.

Among the rarer minerals found in the Amelia County pegmatites, analyses of the garnet, beryl, helvite, allanite, columbite, microlite, apatite, and monazite have been made and are given below in the order named. The general character and mode of occurrence of these minerals in the Amelia County pegmatites have been fully described by Professor Fontaine.[4] No analysis has been made of the zircon. It is much less abundant than in the pegmatites of the Hanover County area and properly belongs to the more sparingly occurring rare minerals in the Amelia County area.

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[2] W. F. Fontaine: Notes on the Occurrence of Certain Minerals in Amelia County, Virginia, *American Journal of Science*, Series 3, vol. 25, p. 332 (1883).

[3] W. F. Fontaine, *ibid.*, pp. 330–339 (1883).

[4] *Ibid.*, pp. 330–339 (1883).

	Spessartite			Beryl
	I	II		III
SiO_2	36.34	35.35	SiO_2	65.24
Al_2O_3	12.63	20.41	Al_2O_3	17.05
Fe_2O_3	.....	2.75	Fe_2O_3	2.20
FeO	4.57	1.75	BeO	12.64
MnO	44.20	38.70	CaO	0.57
MgO	0.47	None	Na_2O	0.68
CaO	1.49	0.94	H_2O	2.70
Ign.	Trace	0.27		
				101.08
	99.70	100.17		
Specific gravity	4.20	.....	Specific gravity	2.702

I. C. M. Bradbury: *Chemical News*, vol. 50, p. 220 (Nov. 7, 1884). See also W. H. Seamon, *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

II. F. W. Clarke: *Bulletin No. 60 of U. S. Geological Survey*, p. 129 (1890).

III. R. W. Barker: Analysis of a Beryl from Amelia C. H., Amelia Co., Virginia, *American Chemical Journal*, vol. 7, No. 3, pp. 175–176 (Oct., 1885).

	Helvite			Allanite
	IV	V		VI
SiO_2	25.48	31.42	SiO_2	32.35
BeO	12.63	10.97	Al_2O_3	16.42
MnO	39.07	40.56	Fe_2O_3	4.49
FeO	2.26	2.99	Ce_2O_3	11.14
Mn	8.66	8.59	Di_2O_3	6.91
S	4.96	4.90	La_2O_3	3.47
Al_2O_3	2.95	0.36	FeO	10.48
CaO	0.71	.....	MnO	1.12
Na_2O	1.01	.....	CaO	11.47
K_2O	0.43	.....	Na_2O K_2O	0.46
	98.16	99.79	H_2O	2.31
				100.62
Specific gravity	......	3.25	Specific gravity	3.323

IV. R. Haines: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882, p. 101.

V. B. E. Sloan: *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

VI. F. P. Dunnington: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882, p. 103.

	Columbite		Microlite
	VII		VIII
Nb_2O_5	31.40	Ta_2O_5	68.43
Ta_2O_5	53.41	Nb_2O_5	7.74
SnO_2	Trace	WO_3	0.30
FeO	5.07	SnO_2	1.05
MnO	8.05	CaO	11.80
CaO	1.27	MgO	1.01
MgO	0.20	BeO	0.34
Y_2O_3(?)	0.82	UO_3	1.59
		Y_2O_3	0.23
	100.22	(Ce, Di)$_2O_3$	0.17
Specific gravity	6.48	Fe_2O_3	0.29
		Al_2O_3	0.13
		Na_2O	2.86
		K_2O	0.29
		F	2.85
		H_2O	1.17
			100.25
		Specific gravity	5.656

VII. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 138 (May, 1882).

VIII. F. P. Dunnington: *American Chemical Journal*, vol. 3, No. 2, pp. 130–133 (May, 1881).

	Apatite		Monazite	
	IX		X	XI
P_2O_5	41.06	P_2O_5	24.04	26.12
CaO	53.94	Ce_2O_3	16.30	29.89
F	3.30	La_2O_3	10.30	26.66
Cl	Trace	Di_2O_3	24.40	
Al_2O_3	0.19	(Y, Er)$_2O_3$	1.10	.....
Fe_2O_3	0.81	SiO_2	2.70	2.85
Insol.	0.63	ThO_2	18.60	14.23
Ign.	0.81	Fe_2O_3	0.90	.....
		Al_2O_3	0.04	.....
	100.74	Ign.	.....	0.67
Specific gravity	3.161			
			98.38	100.42
		Specific gravity	.....	5.30

IX. G. H. Rowan: *Chemical News*, vol. 50, p. 208 (Oct. 31, 1884).

X. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 140 (May, 1882).

XI. S. L. Penfield: *American Journal Science*, vol. 24, pp. 250–254 (1882).

Hanover County Area

The zircon-bearing pegmatites of Hanover County form a part of the recently discovered but fairly well-known rutile area of Goochland and Hanover Counties,[5] which lies about 25 miles northwest of Richmond (see map, Fig. 1). The principal rock of the region is a gneiss of variable composition, chiefly micaceous (biotite and muscovite) and at times hornblendic, cut by numerous pegmatites, some of which are rutile-bearing, and a variety of basic igneous rocks. Microscopic study of thin sections of the gneiss shows it to conform in composition to an original acidic igneous rock of the granite type. The banded structure is secondary, developed by regional metamorphism.

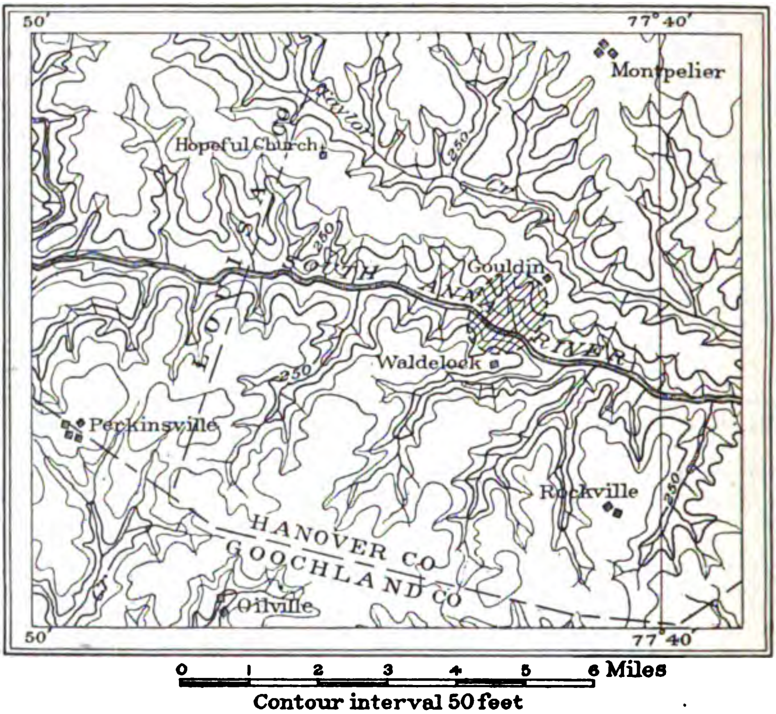

FIG. 1.—TOPOGRAPHIC MAP OF A PART OF HANOVER COUNTY, VIRGINIA, SHOWING LOCATION OF GOULDIN ZIRCON-BEARING PEGMATITE AREA. (BASED ON GOOCHLAND SHEET OF U. S. GEOLOGICAL SURVEY.)

The pegmatite bodies range up to 4 ft. and more in width, are of light color and granitic composition, and lie mostly in the foliation planes of the gneiss. Like the inclosing gneiss, they have been mashed and squeezed from metamorphism and are schistose in structure. Much of the feldspar has been granulated and the quartz mashed into lens-like masses. The pegmatites are dominantly feldspathic, containing both potash and soda-lime varieties, with some quartz, and less muscovite and biotite. Neither amphiboles nor pyroxenes occur in the pegmatites of the Hanover or Amelia County areas. Rutile in small grains and masses up to 15 and 20 lb. and more in weight[6] is an important mineral in some of the Hanover County pegmatites, and is found over parts of the area in such quantity as to be of commercial value.

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[5] T. L. Watson and S. Taber: Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey Bulletin*, III A, pp. 248–261 (1913).

The rutile is an original constituent of the pegmatites and much of it shows granulation and mashing like the fieldspar and quartz. Besides the usual rock-forming minerals of the pegmatites, the rarer minerals are exceptionally few in number. Rutile, the most abundant and important one, is associated with ilmenite which may occur as separate grains and masses or as an intimate mixture or intergrowth with rutile. Occasional apatite has been found, and in places indications of the former presence of pyrite.

In addition to these, zircon has recently been found associated with rutile in the pegmatites near Gouldin in the Hanover portion of the rutile area. (See map, Fig. 1.) The mineral has been found in irregular fragments and masses up to about 2 lb. in weight. One of the larger masses examined by the writer appears to have been broken from a large crystal of the mineral. Like the other constituents of the pegmatites, every specimen of the zircon studied shows mashing and squeezing from metamorphism. The color is irregular even in the same mass, ranging from reddish-brown through grayish to colorless. Although a chemical analysis of the zircon has not been made, laboratory tests carried out on a number of pieces of the mineral show it to be quite pure. The many pieces of the mineral found on the surface, due to the extensive weathering of the pegmatite bodies, encourage the belief that the mineral is by no means a rare constituent of the dikes in this area, and may be found in quantity to be of commercial value.

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[6] F. L. Hess mentions one mass of rutile that was reported to have weighed 200 to 300 lb. *Mining World*, vol. 33, pp. 305–307 (Aug. 20, 1910).

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. Howland, printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

A. H. Lee, Buffalo, N. Y. (communication to the Secretary*).—The statements and results recorded in Mr. Howland's paper can be accounted for by the use of coke with a low sulphur content (0.60 and under). This allows the furnaces to be run on a small volume of acid slag, which it is presumed was done in Mr. Howland's practice. H. A. Brassert has admitted that this was the largest factor in low fuel consumption in his practice. Where the coke contains from 1.00 to 1.25 per cent. of sulphur the situation is quite different and I doubt whether the results mentioned by Mr. Howland could be obtained under such circumstances.

N. M. Langdon, Mancelona, Mich. (communication to the Secretary†).—Gruner's ideal working (which is but a statement of the law first enunciated by Bell) as given in Gordon's *Translation of Studies of Blast Furnace Phenomena*, "supposes that the reduction of the oxide of iron is effected by the CO only, without intervention of solid carbon."

This ideal, if reached, would mean theoretically perfect or 100 per cent. efficiency of reduction. The workings of the large number of furnaces as given in Bell's and Gruner's great works do not show that this standard of perfection ever was reached. This problem has been worked out by the writer for a number of furnaces, and his results, as well as those given by others, all show a failure to reach perfection. Some furnaces, however, approach nearer to perfection than others. The "ideal" as given by Gruner was not stated as a solved problem in blast-furnace practice, but as an ideal to the attainment of which furnace managers should strive. A study of the factors involved from every angle is interesting and instructive and the paper on this subject contributed by H. P. Howland is no exception to the rule. This article is fairly open to criticism, not because of the unique method of presenting the subject, but for the reason that, as the writer conceives, erroneous assumptions are taken which lead naturally to incorrect conclusions.

Referring to the example of two furnaces compared to show the efficiency of the fuel and, for better illustrating the comparison, rearranging the data as follows:

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* Received May 9, 1916.

† Received May 1, 1916.

Exhibit 1

	Coke	Total Carbon	Carbon Used in Iron and Flue Dust	Carbon Burned at Tuyères	Carbon Used for Direct Reduction
1	2,100	1,848	100	1,486	262
1*a*	1,800	1,586	100	1,486	0
2	2,500	2,200	100	1,486	614

1. $\left(\frac{1,486}{1,848 - 100}\right) \times 100 = 85$ per cent. efficiency.

1*a*. $\left(\frac{1,486}{1,586 - 100}\right) \times 100 = 100$ per cent. efficiency.

2. $\left(\frac{1,486}{2,200 - 100}\right) \times 100 = 70.8$ per cent. efficiency.

In the above figures no account has been taken of the carbon required for reduction of the metalloids, which is always direct. The general context of the article, as well as the tables and the one case when the amount of carbon required for the reduction of the metalloids is stated, shows it to be small. Therefore, without vitiating the conclusions to be drawn, it may be neglected.

Case (1) shows 85 per cent. efficiency, (1*a*) 100 per cent. and (2) 70.8 per cent. This is correct for the conditions assumed, but a further examination shows 100 per cent. efficiency of carbon burned at the tuyères and the whole of the inefficiency located at the top or reducing zone. Had the reduction of the ore been 100 per cent. inefficient, that is, had the reduction been made with direct carbon only, it would have required 677 lb. of carbon. Letting *a* equal the amount of solid carbon actually used in the reducing zone, the equation $\frac{677 - a}{677} \times 100$ gives the percentage of efficiency in this zone. Applying to the above we have as the efficiency of reduction:

Exhibit 2

Case 1. $\frac{677 - 262}{677} = 61.3$ per cent.

Case 1*a*. $\frac{677 - 0}{677} = 100$ per cent.

Case 1. $\frac{677 - 614}{677} = 9.3$ per cent.

These percentages are instructive and show that, in order to secure maximum efficiency in the combustion of carbon in the blast furnace, attention should be directed to the reducing zone.

Quoting, "Table 1 shows clearly that, at least in these furnaces, there is no law governing the relation between coke consumption and the per cent. of the carbon which is burned at the tuyères." For comparison and discussion, the data of the two examples 5 and 18, are stated and some additional data inserted.

Exhibit 3

	Coke	Carbon of Coke Gasified in Furnace	Carbon Gasified		Per Cent. Burned at Tuyères	Efficiency of Reduction	Stone Used
			At Tuyères	Reducing Zone			
5	2,198	1,813	1,494	319	82.4	52.9	1,003
18	1,699	1,376	1,155	221	83.9	67.4	804
24	1,623	1,360	1,090	270	80.2	60.1	775
19	1,673	1,384	1,182	202	85.0	70.2	780

While the carbon burned at the tuyères has a direct reference to coke consumption, yet a comparison of percentages is misleading, for the reason that this carbon is only a part of the total carbon burned and this other part, burned in the reducing zone, is variable according to the efficiency of reduction. The fact that a furnace is working with low coke consumption does not by any means indicate that it is even closely approaching Gruner's ideal. This is shown by 24 in Exhibit 3. It will be noted that the coke, total carbon gasified, and carbon gasified at tuyères are lower than 18, while the carbon gasified in the reducing zone is higher, showing a lower efficiency of reduction. The carbon required at the tuyères of any furnace is governed entirely by the quantity, composition and character of the materials entering the furnace and the pig iron produced. The carbon required in the reducing zone depends upon the character of the metallic oxides, the composition of the pig produced and largely upon the degree of efficiency of reduction. From what has been said, it is obvious that if this dictum is accepted, under similar conditions that furnace which shows the highest percentage of efficiency of reduction and thus approaches Gruner's ideal will show the lowest coke consumption, and naturally that is the condition to be desired.

The high coke of No. 5 is due to a lean ore mixture as disclosed by the amount of stone (1,003 lb.) used per ton of iron, necessitating a larger amount of carbon burned at the tuyères than in the case of either No. 18 or No. 24, also to the low efficiency of reduction (52.9 per cent.) in the reducing zone. The lower coke requirement of No. 18, as compared with No. 5, is due to a better ore mixture and higher efficiency of reduction, 67.4 per cent., as compared with 52.9 per cent. The still lower coke requirement of No. 24 is due to a still better ore mixture as disclosed by the smaller amount of stone used, permitting a decrease in carbon required at the tuyères to only 1,090 lb. as compared with 1,155 lb. in No. 18; and this decrease is sufficient to more than overbalance the increase of carbon due to lower efficiency, 60.1 per cent., as compared with 67.4 per cent., in the reducing zone.

For discussion of the work of furnace No. 19, the following tabulation is made:

Exhibit 4

	Coke	Fe	Total Carbon	Carbon in Pig	Carbon Required	
					At Tuyères	Reducing Zone
19 A	1,673	2,031	1,482	98	1,182	202
19 B	1,673	2,031	1,482	98	1,182	36
20	1,658	....	1,464	98	1,182	184

Whether the equation is correct or not as stated in Exhibit 5, column A, the carbon actually used for reduction and the resulting gas is the same as shown in Column B and this latter equation is simpler.

Exhibit 5

		A	B
Reduction by CO	1	$Fe_2+O_3+6CO=2Fe+3CO+3CO_2$	$Fe_2O_3+3CO=2Fe+3CO_2$
Reduction by C	2	$Fe_2+O_3+2C=2Fe+CO+CO_2$	$Fe_2O_3+3C=2Fe+3CO$

The equation 1 A shows 6 CO taken for reduction, only one-half of which was actually used. This unused half, like the bridesmaid at a wedding ceremony, may have performed a very interesting and important part, but, itself, forms no part of the union. This 3CO remains the same after the reaction as before, therefore only 3CO can be considered as having been used, the equation 1 B correctly representing the actual ultimate result.

The reduction of oxide of iron in the blast furnace by solid carbon takes place at a point where the temperature is sufficient in the presence of the surplus of carbon from the fuel to convert any CO_2 which may have formed to CO. The ultimate gas from the reduction by solid carbon is CO only and the ultimate reaction is represented by the equation 2 B.

Reconstructing the summary for "assumption A" which is to illustrate reduction by CO and referring to Exhibits 4 and 5, we have according to equation 1 B the results shown in Exhibit 6.

We see that 166 lb. of solid carbon reduced 517 lb. of Fe, and that 487 lb. of carbon from the total of 1,182 lb. burned to CO at the tuyères reduced 1,514 lb. of Fe. The total CO and CO_2 gases are the same as shown in the original "assumption A." The ratio by volume of $\frac{CO_2}{CO}$ is 0.646, which is very much lower than the ratio 1CO to $1CO_2$ popularly though mistakenly supposed to be the point of saturation by CO_2 at which reduction by CO ceases.

Exhibit 6

		Fe, pounds	C, pounds	CO, pounds	CO_2, pounds
19 A	In pig	.....	98	.....	.....
	Direct reduction of metalloids	.....	36	89	.....
	Direct reduction of Fe	517	166	387	.....
	Indirect reduction of Fe	1,514	487	.....	1,786
	Unused	.....	695	1,622	.....
	Stone	.....	.....	.....	343
		2,031	1,482	2,098	2,129

$$\frac{CO_2}{CO} = \frac{2,129}{2,098} = 1.015 \text{ by weight or } 0.646 \text{ by volume.}$$

Below is "assumption B" reconstructed according to equation 2 B.

Exhibit 7

		Fe, Pounds	C, Pounds	CO, Pounds	CO_2, Pounds
19 B	In pig	.....	98		
	Direct reduction of metalloids.	.....	36	89	
	Direct reduction of Fe	0	0	0	
	Indirect reduction of Fe	2,031	653	.....	2,394
	Unused	.....	529	1,234	
	Stone	.....	.....	.....	343
		2,031	1,316	1,323	2,737
	Carbon saved	.....	166		
			1,482		

$$\frac{CO_2}{CO} = \frac{2,737}{1,323} = 2.069 \text{ by weight or } 1.32 \text{ by volume.}$$

The original "assumption B" is supposed to show the result when "all the CO made at tuyères reduces ore and that the excess carbon is used up by CO_2," as applied to the same data as "assumption A." Assumption B is not tenable because, under the conditions, we must assume the same amount of carbon burned at the tuyères as in assumption A. Now, to reduce the Fe wholly by carbon from CO requires only 653 lb., leaving the balance of 529 lb. of carbon to pass off as CO, and as all of the Fe has been reduced there is an excess of 166 lb. of carbon in the reducing zone which cannot be burned to either CO or CO_2 because there is no oxygen there to unite with it.

The correct statement of the case, assuming the reduction to have been effected with CO only, is shown in Exhibit 7, which shows a saving of 166 lb. of carbon. It may be observed, however, that this saving is probably impossible because of the dilution of the gas by CO_2 which, as shown in the Exhibit, is 1.32. Bell practically states that reduction stops when the proportions are $\frac{CO_2}{CO} = 1.00$.

Assumption G is not well taken as illustrating Gruner's ideal because, in addition to assuming that reduction was by CO only, another unnecessary assumption is injected into the consideration, viz., that 166 lb. more carbon is burned at the tuyères than actually was burned.

Records of heat equations for actual practice of a number of furnaces, collected by the writer, show that for the character of iron given for No. 19, the lowest amount of carbon that can be expected in direct reduction is 156 lb. Assuming this amount for furnace No. 19, we have:

Exhibit 8

		Fe, Pounds	C, Pounds	CO, Pounds	CO_2, Pounds
19 C	In pig	.....	98		
	Direct in metalloids	.....	36[1]	89	
	Direct in Fe	373	120	280	
	Indirect in Fe	1,658	533	.....	1,954
	Unused	.....	649	1,514	
	Stone	.....	.....	.....	343
		2,031	1,436	1,883	2,297
	Saved	.....	46		
			1,482		

$$\frac{CO^2}{CO} = \frac{2,297}{1,883} = 1.22 \text{ by weight or } 0.777 \text{ by volume.}$$

[1] Equal 156 lb.

This shows a ratio $\frac{CO_2}{CO}$ of 0.777 by volume, and is probably as near to Gruner's ideal as could be obtained in actual practice.

The statement that "all the furnaces in Table 1 are doing direct reducing" is correct, while the statement that "all those that are burning less than 1,350 lb. of carbon at the tuyères are not making enough CO to reduce all the Fe_2O_3" is incorrect. It is also incorrect to say that reduction is three times as efficient when done by the direct method. The oxygen from the iron requires precisely the same amount of carbon for its reduction whether taken from the solid carbon of the fuel or from the carbonic oxide of the gas. When the carbon can be taken from the carbonic oxide produced by the fuel necessarily burned at the tuyères, it obviously saves an equal amount of carbon from the fuel that, otherwise, would be needed and, therefore, is more efficient than solid carbon. The extent to which this can be done is the measure of the approach to the ideal.

All furnaces do not reach Gruner's ideal. In fact, it is very doubtful if any do or can. Some, however, come nearer to it than others. They may show very low coke consumption and yet be far from reaching the ideal. The reason for this is not because of a violation of or considerable variation from Gruner's ideal, but because of some other favorable condition. This is shown in Exhibit 9.

Exhibit 9

	Total Carbon Gasified, Pounds	Carbon Gasified at Tuyères, Pounds	Carbon Gasified in Reducing Zone Pounds
Average of first 10 furnaces	1,800	1,522	278
Average of last 16 furnaces	1,384	1,146	238
Difference	416	376	40

It will be noted that the last 16 furnaces show a decrease of 416 lb. of carbon required, and a decrease of 40 lb. in the reducing zone due to a nearer approach to Gruner's ideal. The great decrease of 375 lb. is in carbon burned at the tuyères evidently due to a better ore mixture, requiring less flux, as shown by the stone used. Less slag is produced, therefore less carbon is required to be burned at this point.

From what has preceded, it will be evident that the factors governing coke consumption will be found in the quantity and quality of the materials entering the furnace, coke, ore, limestone and blast; also in the method of reduction—that method tending to the highest economy, which approaches nearest to Gruner's ideal.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Motor Truck Operation at Mammoth Collins Mine, Shultz, Ariz.

BY WILBERT G. McBRIDE,* B. S., LONG BEACH, CAL.

(Arizona Meeting, September, 1916)

Two Alco 3½-ton motor trucks were used by Young Bros. while operating at the Mammoth Collins mine at Shultz, Ariz. One was equipped with an oil tank holding 1,075 gal. and was used for the transportation of "tops." The other was fitted with a stake body and used to carry machinery, wood, rails, pipe and all classes of miscellaneous supplies. The bodies were made of oak with maple flooring and were attached to the frame of the chassis by U-bolts, to avoid drilling the main members of the frame.

Most of the hauling was done from Tucson, a distance of 47½ miles. During the first 3 months, part of the road was in very bad condition and the tire cost was excessive. After this part was repaired the road was in fair condition, but never good. There were no excessive grades or bad sand, but wagon ruts, too narrow for the truck wheels and of a different gage, caused heavy tire loss; while chuck holes, sharp curves and stones, both imbedded and loose, were objectionable features. During wet weather the trucks could not get sufficient traction to climb some of the hills and were likely to stick in the mud in certain places, so that no attempt was made to run them unless they were on the road when the rain started. This lost time amounted to about 5 per cent. of the total, but, whenever possible, it was utilized in making minor repairs.

The price of gasoline was from 17 to 21 c. per gal. Rubber tires were used throughout. Drivers were paid $4.50 to $5 per shift, and a return trip to Tucson was counted as two shifts even when made in one day. Drivers were provided with a room in Tucson and were paid for all time lost due to causes beyond their control. Trucks were loaded one way only.

Speedometers were placed on both trucks but the excessive vibration soon caused them to fail. For this reason, and because no account was taken of the distance covered in picking up a miscellaneous load or in other minor ways, the mileage given is under the actual distance traveled. Some of the weights had to be estimated, but care was taken to have the number of ton-miles low rather than high, to avoid underestimating the costs. The cost of hauling from Tucson to the mines was $12 per ton

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[*] Consulting Engineer.

with the trucks, while the best possible team price was $15. Teams made one return trip a week, while the truck regularly made one in two days and could always, and many times did, do it in one day. The loss of time due to wet weather would be about half as much with teams as with trucks.

The table of detailed costs given below covered the period from Aug. 21, 1913, to Aug. 15, 1914, the only time in which the trucks were continuously employed. From Aug. 15, 1914, to March 30, 1916, the trucks were used intermittently, but the figures for this period have been excluded as not being representative. If included, they would somewhat lower the cost per ton-mile. Just prior to the close of the period covered by the cost figures, the trucks were overhauled and put in good condition; new rear wheels were put on and new tires secured. The cost of all this was charged to operation. Allowance for extra tires on hand would reduce the cost per ton-mile approximately ¾ c., leaving a net cost of about 25c. With loads on the return trip this cost per ton-mile would be lowered at least 40 per cent.

Motor Truck Operating Data

Total distance traveled by trucks	23,000 miles
Total work done by trucks	42,700 ton-miles
Average distance covered per gallon of gasoline	4.5 miles
Average distance covered per gallon of lubricating oil	128 miles
Average speed, loaded	7 miles per hour
Average speed, light	7.8 miles per hour

Details of Costs

	Total Cost	Per Cent. of Total	Per Truck-mile	Per Ton-mile
Wages of drivers	$2,623.32	23.91	$0.1141	$0.0614
Wages of helpers	286.50	2.62	0.0125	0.0067
Repairs, labor	581.74	5.30	0.0253	0.0136
Repairs, lost time	156.15	1.42	0.0068	0.0037
Oils, grease and waste	379.17	3.46	0.0165	0.0089
Gasoline	1,610.49	14.68	0.0700	0.0377
Tires	2,445.75	22.30	0.1063	0.0573
New parts	515.08	4.69	0.0224	0.0121
Miscellaneous supplies	348.82	3.18	0.0152	0.0082
Incidental expense	226.21	2.06	0.0098	0.0053
Depreciation	1,796.80	16.38	0.0781	0.0421
Total	10,970.03	100.00	0.4770	0.2570

The advantages of the motor truck over the team and wagon are many—increased speed, ability to work 24 hr. per day when necessary, and lower cost on long hauls—but its adoption by the mining industry has been slow. Where trucks are used around mines they are usually driven by cheap, inexperienced men, the upkeep and repairs being turned over to the regular mine mechanics. It would be equally good practice to employ a timber framer to make a dining-room table. Just as the niceties of cabinet making are unknown to the timber framer, the exact adjustments and fine workmanship of the high-speed engine and transmission gears of a motor truck are beyond the ken of the mine mechanic, one of the least skilled of his class. If there are enough motor vehicles at the mine, the master mechanic probably turns the work over to one or two men who, in time, become indifferent auto-mechanics, but in the meantime the cost of maintenance soars and often the trucks are condemned. The aim of the makers of all motor vehicles is to secure the maximum of strength and power with a minimum of weight and size. To do this, high-speed engines, the best of materials and the finest of workmanship are employed and parts are reduced to the least possible weight consistent with strength and durability. This is just the reverse of the ordinary American mechanical practice in which reliability is secured by slow speed and large size, the amount of material used and the space occupied being minor considerations. It is, therefore, unreasonable to expect the mechanic trained in one school to understand immediately and adapt himself to the ways of the other. It must also be remembered that no other machine is given the hard use and necessary abuse that a motor truck receives. The road vibration, alone, will loosen nuts and rivets which, if not attended to in time, will cause serious trouble. Where only one or two trucks are used, the drivers should be competent mechanics and should be held responsible for the maintenance of their machines. Where several are used, they should be under the direct supervision of a thorough truck mechanic who is held responsible for operation and given entire control of the drivers and repair work. His constant care will detect and remedy many incipient defects and prevent expensive and annoying breakdowns. With the exception of the time required for periodic overhauling, he should be able to keep the trucks in almost continuous service. This will make possible the employment of cheaper drivers without undue damage being done to the machines.

Motor trucks should not be installed without careful consideration of the roads to be traveled. The difference between the cost of motor truck and team hauling is largely controlled by the quality of the road, and on really bad roads the motor truck is decidedly the more expensive. Many roads are fatal to truck haulage, and considerable experience is required to decide this question without an actual test of some duration. Excessive grades are to be avoided, especially long ones. The ordinary truck will pull over a short 20 per cent. grade with ease, but will give great trouble on a long one of half that rise unless special cooling arrangements are made. Grades greatly increase the tire and gasoline consumption and decrease the life of the machine. Rocky roads, particularly when the rocks are sharp or loose, are very hard on tires. Deep sand is difficult to cross, and for this class of road the caterpillar tractor and the four-wheel-drive truck have distinct advantages. Trucks which drive on the rear wheels only cannot operate in heavy sand. Narrow or rutted roads are objectionable for the larger-sized trucks because they throw all the weight on one of the rear dual tires from time to time, and this overloading is injurious to the rubber. Fairly deep streams can be crossed, but mud is an absolute barrier except to the caterpillar type of tractor. Few dirt roads will stand up under a 7-ton truck, but those of 4 tons, or under, do less damage than the ordinary freight wagon.

Unfortunately, trucks are not designed to suit mining conditions. At Shultz we found it necessary to cut down the gear ratio, increase the size of the wheels and tires and add bumper or auxiliary springs. Had the grades been steeper it would have been necessary to increase the cooling capacity.

For long hauls the motor tractor will probably replace the motor truck. It will operate at a lower cost because the load will be carried on iron tires, and, as the table of detailed costs shows, the rubber tires account for 22.3 per cent. of the total. Tractors travel more slowly than motor trucks, but the tonnage hauled in a trip is much greater. They are also easier on roads, as the load is distributed over several trailers. By using extra trailers, loading and unloading can be done while the tractor is on the road.

The make of a truck is not as important as the care it receives. Almost any standard make will do good work if given careful attention, but none will be satisfactory if not well cared for. Economy should not be sought in the lubricants used; the best oil is none too good for a motor truck. Overloading should be scrupulously avoided. A truck may be made to carry many times its rated load without breaking down but the damage is none the less real because not immediately apparent. High speed, particularly if the road is rough, should be avoided, since it subjects the machine to excessive strains and vibration. Most trucks are now equipped with speed governors, but these are easily tampered with and must be carefully watched. When they are not used, the drivers should be carefully instructed as to the speed limits and compelled to respect them.

Distillate and "tops" are now successfully used on trucks, by the application of a special carburetor. Their use should effect a material saving in the gasoline cost which now amounts to almost 15 per cent. of the total. "Tops" usually sell for 30 to 35 per cent. and distillate for 50 to 60 per cent. of the price of gasoline. With a properly designed carburetor, the available power in the lower-grade fuel will be about the same as in the gasoline, but the carbon deposition will probably be somewhat greater.

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Trade Catalogs

To a majority of the public, the name DuPont suggests military powder—munitions of war. The *DuPont Products Book* A-430 (April, 1916) lists 251 distinct commodities made and sold by this company, many of which are far removed from destructive work.

Mining men are fully cognizant of the need for explosives in constructive work, but they may be surprised to know that side by side with the powder and dynamite this company manufactures such peaceful articles as substitute leather, combs and collar buttons.

The substitute for leather is *Fabrikoid*, varying in weight from a quality light enough for bookbinding to one heavy enough for trunks and automobiles, and the combs and collar buttons (with scores of other articles) are made from *Pyralin.*

The book also contains a long list of chemicals, and a shorter list of special products and miscellaneous materials such as charcoal, wood flour and saltpeter.

The Jeffrey Manufacturing Co. of Columbus, Ohio, making elevating, conveying, crushing, screening and power transmission machinery, has just issued its latest Bulletin, No. 177, featuring self-propelling wagon and truck loaders for handling crushed stone, sand, gravel, clinker, coke, etc.

These machines are described as having a capacity of 1 to 1½ cu. yd. of material per minute, with power to load crushed stone maximum size pieces through 3½-in. ring and to load trucks or wagons in 3 or 4 min.

The self-propelling device is designed to enable one man to operate the loader as it is fed into the pile, and to move the machine about under its own power.

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These loaders are driven by 5 hp. induction motor or 6 hp. gasoline engine. The type "A-15" loader is built with a collapsible elevator, intended to make it available about all parts of the yard and easier to operate over rough ground, and also to permit it to be used where low sheds, trestles and other overhead obstructions are encountered.

The bulletin contains illustrations of many interesting installations, specifications, prices and complete details of their different types of loaders.

The Ingersoll-Rand Co. has recently issued three new bulletins:

Form 9023, 24 page, 6 × 9, on "Imperial" Tie Tamping Outfits. This apparatus is said to have had a gradual development under actual working conditions on some of the largest railroad systems in the country, extending over a period exceeding two years; it is comparatively new to the trade, but the manufacturers feel that the railroads have been quick to appreciate the economy attendant upon the use of one or more of these outfits. The Tamper is designed principally for use in the operations of laying new track, replacing old track and surfacing existing track, but may be employed for other purposes. It is said to produce equally effective results in any kind of ballast—stone, gravel, earth, cinders, chat, slag, etc.

Form 3026, 40 page, 6 × 9 catalog on Ingersoll-Rogler Class "PRE" Duplex Direct-Connected Electrically Driven Air Compressors. Among the principal features of design may be mentioned the "Ingersoll-Rogler" Valve, the "Clearance Controller," Direct-Connected Drive, Improved Intercooler and Auxiliary Water Separator. Catalog is printed in two colors and profusely illustrated to show construction details. Four pages are devoted to pressure charts and tables showing sizes and capacities.

Form 3312, 20 page, 6 × 9 catalog on Imperial "XB" Duplex Power-Driven Air Compressors. Illustrated.

Electric "Industrial Klaxons," made by the Lovell-McConnell Mfg. Co. and distributed by the Western Electric Co., are pictured and described in a pamphlet advocating their use as fire alarms and call signals. They are now being used on moving cranes to warn the workmen to stand from under, and are suggested for use in mines and quarries as time, blast, and elevator signals; for use on electric locomotives, underground telephones, etc.

From the Western Electric Co., also, come three bulletins telling of "Square D" switches and motor starters made by the Detroit Fuse and Mfg. Co. These articles bear the Fire and Accident Label of the Underwriters' Laboratories, Inc.

Bulletin No. 37 deals with Steel Enclosed Switches; Bulletin No. 40 with Ironclad Fused Switches, Plunger Type; and Bulletin No. 41 with Induction Motor Starters. Specifications, list prices and dimensions are given.

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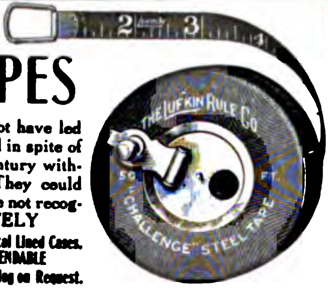

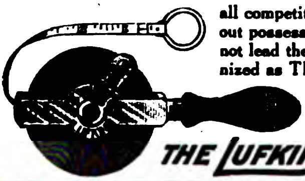

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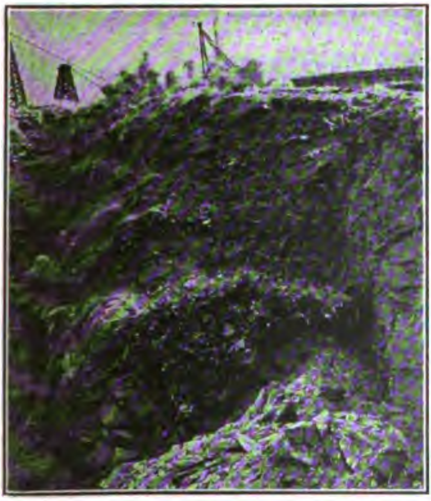

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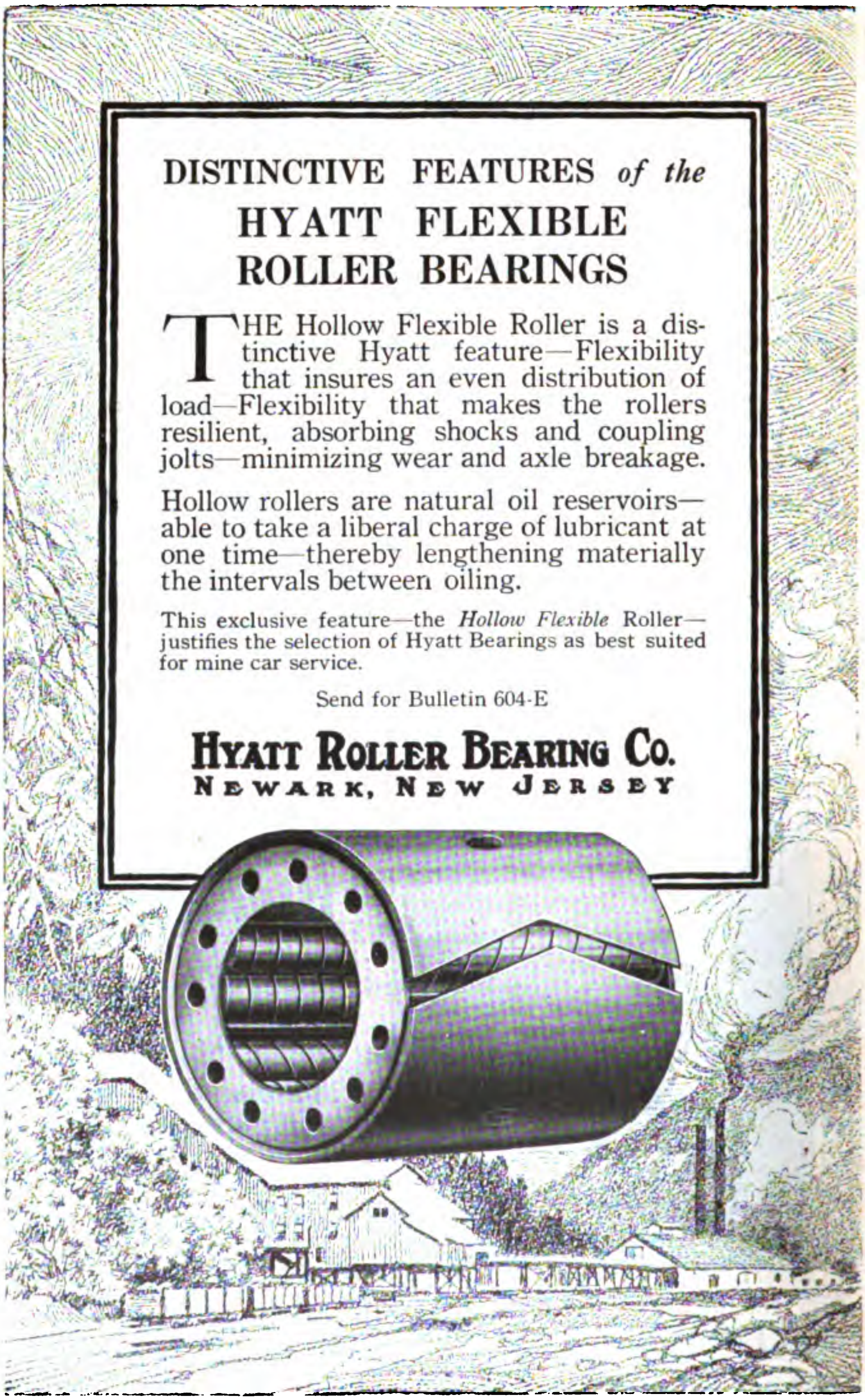

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Roebling Wire Rope

gives it a distinction which causes it to be preferred by discriminating buyers.

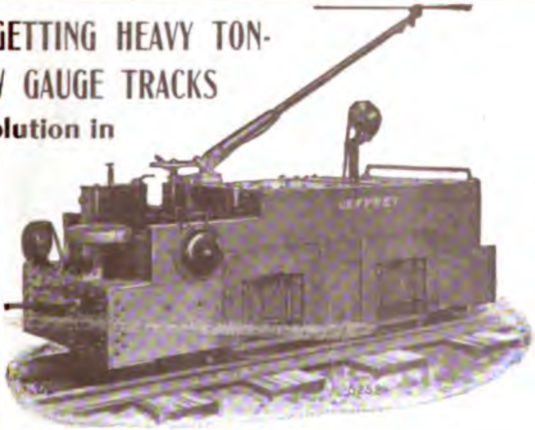
All metals are subjected to rigid tests; the wire is drawn from these metals at the Roebling works, and the stranding is done upon designs and by shop methods developed from an experience of 77 years spent in making wire rope.

John A. Roebling's Sons Company
Trenton, N. J.

THE PROBLEM OF GETTING HEAVY TONNAGE OVER NARROW GAUGE TRACKS

Finds a Ready Solution in

JEFFREY
ELECTRIC HAULAGE LOCOMOTIVES



Spending money to widen rock tunnels to increase haulage capacity is not necessary, as Jeffrey Electric Trolley and Storage Battery Locomotives will take out the tonnage on gauges as low as 18″.

Send for Special Bulletins No's 111-2 and 107-2 on the narrow gauge question. They describe installations that prove conclusively that high tonnages are not restricted to wide gauges.

THE JEFFREY MFG. CO., 902 N. Fourth St., Columbus, O.
New York Boston Philadelphia Pittsburgh Milwaukee Chicago Birmingham
St. Louis Denver Montreal Seattle

Scores of G-E Motors Drive Mills for the Calumet and Hecla Mining Company

A remarkable installation of 64 electrically driven Hardinge Mills at the Calumet and Hecla Mining Company must operate 24 hrs. a day for long periods.

For this, the largest installation of its kind in the world, G-E 40 H. P. squirrel cage induction motors were selected. These motors have a high efficiency—so important for long period operation—and the required high starting torque.

Practically standard construction was selected for these motors, and the selection has been justified during the last three years by an operating record which has been satisfactory in every respect.

These motors are so dependable in operation that but two men per shift are required for operating the 64 mills and motors.

Hundreds of grinding mills are driven by G-E motors, the capacity of which in some cases is as large as 225 H. P. Among those using these motors are the Anaconda Copper Mining Company, the Inspiration Consolidated Copper Company and others equally well known.

General Electric Company

General Office: Schenectady, N. Y. District Offices in:

Boston, Mass. New York, N. Y. Philadelphia, Pa. Atlanta, Ga.
Cincinnati, Ohio Chicago, Ill. Denver, Colo. San Francisco, Cal.
St. Louis, Mo.

Sales Offices in All Large Cities

6244

38,285

AUGUST, 1916

For Explanation of Reference Letters See Page xi

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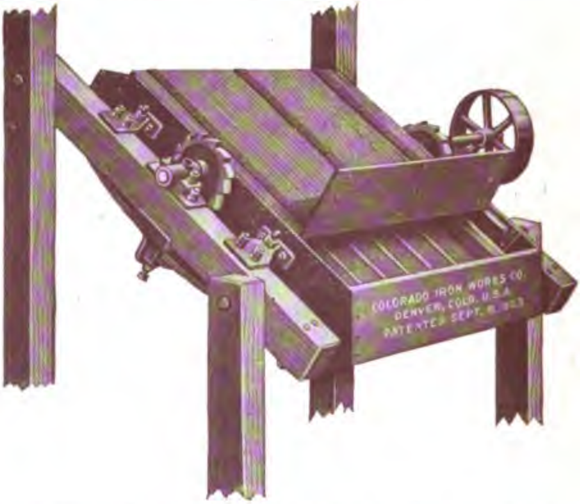
C.

COLORADO IRON WORKS COMPANY

ORE SMELTING EQUIPMENTS **ORE MILLING MACHINERY**

The Impact Screen

is a screen which wins out in competition with the best of them. That it holds its own is not the whole truth, for it displaces others. It is a superseder, getting into plants where others were first installed, and when it is in, it is in to stay. Real worth has earned it the place at the top and it has no close second.


COLORADO IRON WORKS CO.
DENVER, COLO. U.S.A.
PATENTED SEPT.

Don't be misled by a slight resemblance in appearance. The particular jolt and the particular direction of that jolt keep the meshes cleaner than any other, and attempts to avoid the patented features which bring about these results have ended in failure.

May we tell you more about it?

Colorado Iron Works Co.

Denver, Colorado

Main Office and Works: 33rd and Wynkoop Sts. Down Town Office: 206 Ideal Building

AMERICAN INSTITUTE OF MINING ENGINEERS
29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. ____
(Name in Full)

Occupation ____

Address ____

is hereby proposed by the undersigned, as a ________________

____

of the American Institute of Mining Engineers.

____
____
____
} **Signatures of three Members or Associates.**

Place of birth ________________________ *Year of birth* ________

Education, general and technical, when, where and how acquired, with degrees, if any.

Dates	

Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)

Dates	

Present position: ____________________

____

(If the applicant has previously been proposed as a Member, Associate or Junior Member of the Institute, this fact should be stated.)

I agree, if elected, to accept election within four months, and to abide by the Constitution and By-Laws

Signature of Applicant: ____________________

Dated ____________________ *191*

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—Members.

Sec. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

Sec. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

AUGUST, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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York, Pennsylvania, under the Act of March 3, 1879.

TRANSACTIONS FOR 1916

Vol. 51 has been sent to all members who have paid their 1916 dues. Vols. 52 and 53, the last two volumes for 1916, will be shipped about the middle of August to all except those owing current dues. Members who do not receive them within a reasonable time are asked to notify the Secretary.

Back volumes to complete sets may be purchased from the Institute at reduced rates.

Arizona Meeting, Sept. 18 to 25, 1916.

ARIZONA MEETING, SEPTEMBER 18 to 25, 1916

For the first time in its history the Institute will hold a meeting in the State of Arizona.

A few years ago Arizona stood third in the copper-producing districts of the United States. Since that time, with the development of porphyry mines, the output has gone up with great rapidity until it not only is the leading district, but its output at the present time is at the rate of nearly double the Montana output, which now stands second in the list.

In connection with this development, most interesting departures have been made in methods of mining, concentration and smelting. Principles of mining and metallurgy of the greatest interest will be covered by papers which will appear in the *Bulletin* and be presented at Technical Sessions, while ample opportunity will be afforded for those who go to the meeting to see all the points of interest to them. In addition, members will be well rewarded by the marvelous natural features to be seen on the trip, including the petrified forests, the cliff dwellings on the Apache Trail and the Grand Canyon of Arizona.

It is planned to start the special train from New York, but members and guests may join the special at other points en route as indicated by the following schedule:

ITINERARY

Thursday, September 14th

Lv	New York, Grand Central Terminal		5.30 p. m.
	Dinner in dining car a la carte		

New England Members via B. & A. R. R.

Lv	Boston	(Regular train)	2.00 p. m.
Lv	Worcester	(Regular train)	3.12 p. m.
Lv	Springfield	(Regular train)	4.37 p. m.
	Dinner in Dining Car a la carte		
Ar	Albany	New York Central R. R.	7.45 p. m.
Lv	Albany		9.00 p. m.
Lv	Schenectady		9.32 p. m.
Lv	Utica		11.08 p. m.

Friday, September 15th

Lv	Syracuse	12.30 a. m.
Lv	Cleveland	7.25 a. m.
Lv	Toledo	10.05 a. m.
Ar	Chicago	4.00 p. m.
Lv	Chicago, C. R. I. & P.	8.05 p. m.

Saturday, September 16th

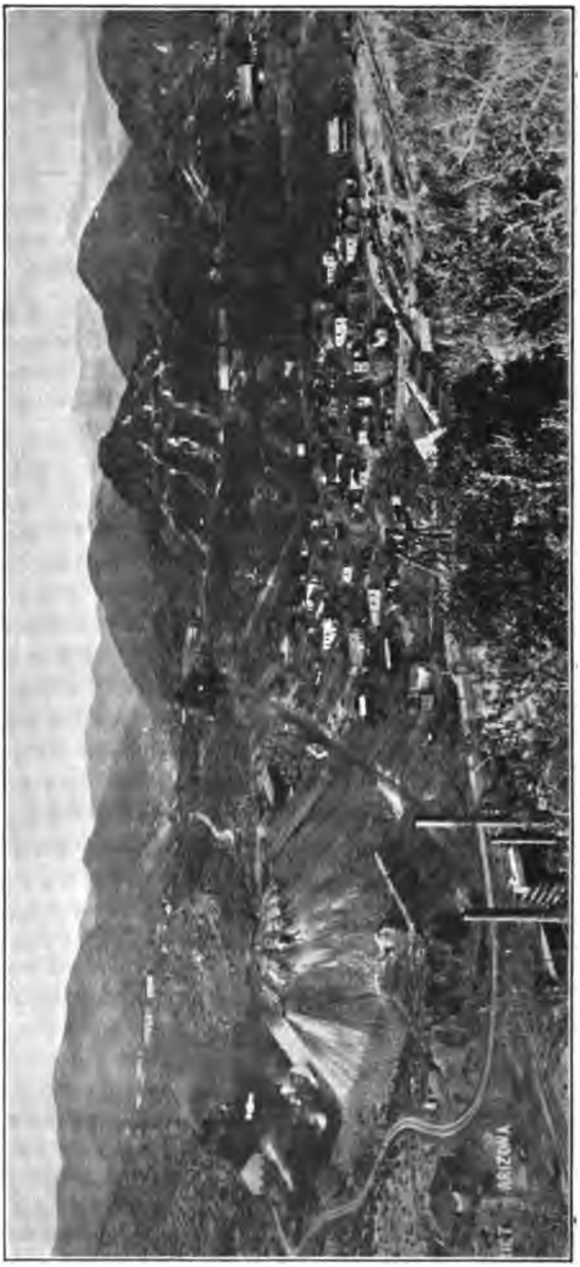
Lv	Kansas City, C. R. I. & P.	11.00 a. m.
Lv	Topeka, C. R. I. & P.	1.05 p. m.
Lv	Hutchinson	6.05 p. m.

Sunday, September 17th

Lv	Santa Rosa, E. P. & S. W. R. R.	6.30 a. m.
Ar	El Paso	2.40 p. m.
Lv	El Paso, Southern Pacific	10.20 p. m.

Monday, September 18th

Ar	Santa Rita	4.00 a. m.
	Visit mines of Chino Copper Co. and mill of Empire Zinc Co.	
Lv	Santa Rita, Sante Fe.	2.00 p. m.



BISBEE DISTRICT, ARIZONA, SHOWING SHAFTS OF THE CALUMET & ARIZONA AND COPPER QUEEN MINING COMPANIES.

A HOPI INDIAN VILLAGE IN ARIZONA. (Bulletin 613, U. S. Geological Survey.)

Av Hurley... 2.45 p. m.
Visit mill of China Copper Co.
Lv Hurley, Sante Fe............................... 10.15 p. m.

Tuesday, September 19th

Ar Douglas... 7.00 a. m.
Visit reduction works of Copper Queen Consolidated Mining Co., and of Calumet & Arizona Mining Co.
Technical Session on "Smelting."
Technical Session on "Leaching."
Lv Douglas, E. P. & S. W.......................... 11.00 p. m.

Wednesday, September 20th

Ar Bisbee, Arizona................................. 1.00 a. m.
Visit to mines of district.
Technical Session, subject "Mining and Geology"
Lv Bisbee, E. P. & S. W............................ 10.00 p. m.

Thursday, September 21st

Ar Globe... 9.00 a. m.
Visit to mines and reduction works of Old Dominion Copper Co.
Technical Session, subject "Concentration and Flotation."
Technical Session, subject "Fine Grinding."

Friday, September 22nd (Globe)

Visit reduction works of International Smelting & Refining Co., and mills of The Inspiration Consolidated Copper Co., and the Miami Copper Co.

Technical Session, subject "Mining and Smelting."

(The Ray Consolidated Copper Co. will welcome members and guests who desire to visit the mine at Ray. Special arrangements must be made by omitting other features on the program and with the understanding that those who go to Ray must reach Phoenix in time to take the train from there.)

Saturday, September 23rd (Globe)

Lv Globe by automobile over Apache Trail, including visit to Roosevelt Dam.
Ar Phoenix about.................................. 5.30 p. m.
Lv Phoenix, Sante Fe.............................. 8.00 p. m.

THE APACHE TRAIL

History records that the ancient trail leading to the ruined cliff dwellings in the Salt River Valley was discovered in 1540 by Francisco Vasquez de Coronado, the Spanish explorer, in his search for the treasures of the "seven cities" of Cibola, the abode of the Zuni Indians—the "Fathers of the Pueblos." Later the trail was used by the Apache Indians and this wild and rugged region was one of their last retreats—the scene of desperate encounters with our troopers in the campaign of 1872. Today the modern motor car follows the same historic and romantic trail through scenery of unusual variety and beauty over the highway between Globe and Phoenix, a distance of 120 miles.

The building of the great Roosevelt Dam, which required the construction of the roadway through the wildest and most rugged sections of the route, has made possible this unique mountain tour, and the consequent transformation of the Salt River and Tonto Creek basins into a great lake region has added a charming aspect to views already grandly impressive.

THE GRANITE GORGE IN THE GRAND CANYON, NORTHWEST OF GRANDVIEW POINT. (Bulletin 613, U. S. Geological Survey.)

CLIFF DWELLINGS

At a point four and one-half miles east of Roosevelt, a wagon trail leads up a small canyon to a point where a foot trail may be taken up the steep mountain side to a group of ancient cliff dwellings which are well worth visiting. Another and more extensive group of ruins can be reached two miles farther by trail.

ROOSEVELT DAM

This great dam has a maximum height of 280 feet. When the reservoir is full the water backs up fifteen miles in the Tonto and fifteen miles in Salt River. A hydro-electric plant located below the dam furnished current for use during the construction period and now provides power for irrigation pumping on the Salt River Project and for sale to private consumers.

A WATER BOTTLE IN THE DESERT.
Taking a drink pressed from the pulp that forms the interior of a barrel cactus, or visnaga. (Bulletin 613, U. S. Geological Survey.)

The Salt River Valley is a land of perennial summer, of oranges, of cotton and of dates. Alfalfa is a staple, and ostrich farming is among the successful enterprises. Irrigation has hastened the valley's progress and confirmed its business stability. Phoenix is its metropolis, an energetic and attractive city, and the capital of the State.

Sunday, September 24th

Ar Grand Canyon	8.00 a. m.
Lv Grand Canyon, Sante Fe	7.40 p. m.

GRAND CANYON OF ARIZONA

The series of tremendous chasms which form the channel of the Colorado River in its course through northern Arizona, reach their culmination in a chaotic gorge, 217 miles long, from 9 to 13 miles wide, and more than 6,600 ft. below the level of the plateau.

Standing upon the brink of the plateau at the point of the canyon's greatest width and depth, the beholder is confronted by a scene whose majesty and beauty are indescribable. It is one of the few widely advertised spots which one need not fear approaching with anticipations too exalted. It has never been adequately described, and never will be.

The picture on the cover of this *Bulletin* is of the north side of the Grand Canyon as viewed by telescope from El Tovar Hotel. This illustration and others of Arizona scenery herewith reproduced are from *Bulletin* 613, *U. S. Geological Survey,* Guidebook of the Western United States, Part C, The Santa Fe Route. These cuts were obtained through the courtesy of Dr. George Otis Smith, Director. The letters indicate rock formations as follows:

G, granite and gneiss; *U,* sandstone, red shale, and limestone (Unkar); *T,* sandstone of Tonto Platform; *Sh,* shale of Tonto group lying directly on quartzite of Unkar, *R,* limestone (Redwall); *S,* red sandstone and shale (Supai); *C,* gray sandstone (Coconino); *K,* limestone (Kaibab). The Redwall butte in center is Cheops Pyramid. Beyond it are Buddha and Manu temples. The background is the Kaibab Plateau.

Monday, September 25th

Lv Albuquerque, Santa Fe	7.00 p. m.

Tuesday, September 26th

Lv La Junta, Sante Fe	7.10 a. m.
Lv Kansas City, Sante Fe	11.00 p. m.

Wednesday, September 27th

Ar Chicago	11.15 a. m.
Lv Chicago, New York Central	12.40 p. m.
Lv Cleveland, New York Central	7.35 p. m.

Thursday, September 28th

Ar Albany	6.27 a. m.
Ar Boston	11.55 p. m.
Ar Grand Central Terminal	9.40 a. m.

Fares quoted below include railroad and sleeping car fares for the entire trip. Fares from New York and Albany also include excess fares applying to Chicago and return owing to extra fast schedule of train.

For the exclusive use of a compartment by one person, one and one-half railroad tickets are required and for a drawing room, two tickets are required.

The arrangements as announced, also railroad and sleeping car fares shown below, are conditional upon one hundred fares New York to El Paso, one hundred twenty-five fares El Paso to Grand Canyon and one hundred fares Grand Canyon to New York.

Starting point*	Lower 1 Person	Upper 1 Person	Compartment 2 Persons (Each)	Drawing Room 2 Persons (Each)
New York	$228.50	$217.50	$233.50	$261.00
Albany	216.60	205.60	221.60	249.10
Schenectady	208.96	197.96	213.96	241.46
Utica	205.84	194.84	210.84	238.34
Syracuse	203.84	192.84	208.84	236.34
Rochester	200.60	189.60	205.60	233.10
Buffalo	198.40	187.40	203.40	230.90
Cleveland	188.80	177.80	193.80	221.30
Toledo	186.30	175.30	191.30	218.80
Chicago	176.80	165.80	181.80	209.30
Kansas City	160.30	149.30	165.30	192.80

APPLICATIONS

Those desirous of joining the Tour should promptly communicate with Mr. W. V. Lifsey, Assistant General Passenger Agent, New York Central Lines, 1216 Broadway, New York City. Reservations will be made in the order in which applications are received.

Members of the Arizona Section desiring accommodations on the Special Train should communicate with Mr. Arthur Notman, Box 400, Bisbee, Ariz.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JUNE 23, 1916

Report was made by the Committee of National Engineering Societies on the Reserve Corps of Engineers and it was voted that a committee of three be formed—consisting of the President of the Institute, Dr. Henry S. Drinker and the Secretary of the Institute—to select members in different parts of the country to advise with the local U. S. Engineer Officers as to qualifications of applicants for the Reserve Corps of Engineers.

NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President.

Two officers, known as Director and Vice-President.

Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York City and district, which is provided for in the Constitution. (N.B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.

District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.

District No. 4. - Minnesota, Wisconsin, and Michigan.

District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.

District No. 6. California and Nevada.

District No. 7. Utah, Colorado, Arizona, and New Mexico.

District No. 8. Louisiana and Texas.

District No. 9. Other Southern States and District of Columbia.

District No. 10. Mexico.

District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, L. D. Ricketts (not eligible for re-election).

Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
Chairman, Committee on Nominations.

ENGINEERS' RESERVE CORPS

The Army Reorganization Act, which became effective July 1, 1916, contains a provision authorizing the organization of an Officers' Reserve, in which will be included a Corps of Reserve Engineers. There will also be an Enlisted Reserve of skilled workmen in technical lines. The special provisions of the bill relating to the Engineer Officers' Reserve were formulated with the help of the Joint Committee of the several national engineering societies, which has been at work on the matter for over a year with the War Department and Committees of Congress.

Recognizing that engineering science is the basis of success in modern warfare, it is apparently the purpose, in thus mobilizing the engineering talent of the country, to give every engineer or technical specialist an opportunity, if the country can use his services, to do that which he can do best, and not necessarily to try and train all to the same standard of military engineering work for service with troops. At the same time, exceptional opportunities are offered to those who wish to qualify for this kind of active service and for those engineers who are inclined to seek commissions as engineer officers in the permanent establishment. It will be noted that the usual age limits are waived in the case of the Engineers' Reserve Corps.

But for the disarrangement of the ordinary routine of the War Department, resulting from the present emergency on the Mexican border, it is probable that the machinery for organizing the Engineers' Reserve Corps would already be in motion, but under the circumstances, it will be necessary to await the convenience of the authorities.

The report of the Joint Committee follows:

To the Boards of Direction of the

AMERICAN SOCIETY OF CIVIL ENGINEERS,

AMERICAN INSTITUTE OF MINING ENGINEERS,

AMERICAN SOCIETY OF MECHANICAL ENGINEERS,

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS,

AMERICAN INSTITUTE OF CONSULTING ENGINEERS.

DEAR SIRS:—On December 22, 1915, the Joint Committee (consisting of the Chairmen of your several committees), formed under the authority of the five societies in order to facilitate the carrying out of the organization of an Engineer Reserve as part of the military forces of the United States, reported the progress that had been made up to that time in the matter of the necessary legislation.

The Joint Committee has now to report to you that the legislation referred to is embodied in a bill which recently passed Congress, has been signed by the President, and will become effective July 1, 1916. This bill, known as the Army Reorganization Act of 1916, contains provisions for the organization of an Officers Reserve, including engineers. An abstract of the principal clauses relating to the Officers Reserve and to the Enlisted Reserve Corps is appended hereto.

The War Department is now formulating the plan of organization of the Reserve Corps of Engineers and of the Enlisted Reserve Corps and the requirements that will be prescribed for enrollment therein. As soon as practicable after July 1, the War Department will issue an invitation to the engineers of the country to apply for commissions in the Reserve Corps of Engineers, and there will be available for those interested full and complete information as to the requirements for commissions in the several grades from Second Lieutenant to Major, and for Master Engineers and Sergeants, and the method of procedure to secure commissions or enlistment.

The Joint Committee has notified the War Department that its services in these matters are at the call of the Department.

Very truly yours,

WM. BARCLAY PARSONS, Chairman,
American Society of Civil Engineers,
H. S. DRINKER,
American Institute of Mining Engineers,
WM. H. WILEY,
American Society of Mechanical Engineers,
B. J. ARNOLD,
American Institute of Electrical Engineers,
RALPH D. MERSHON,
American Institute of Consulting Engineers.

23D JUNE, 1916.

ABSTRACT

THE ENGINEER SECTION OF THE OFFICERS RESERVE CORPS AND THE ENLISTED RESERVE CORPS

From the Army Reorganization Act of 1916, Sections, 37, 38, 39, 55

Sec. 37.—The Officers Reserve Corps.—For the purpose of securing a reserve of officers available for service as temporary officers in the Regular Army, as provided for in this Act and in Section eight of the Act approved April twenty-fifth, nineteen hundred and fourteen, * * * there shall be organized, under such rules and regulations as the President may prescribe not inconsistent with the provisions of this Act, an Officers Reserve Corps of the Regular Army. Said corps shall consist of sections corresponding to the various arms, staff corps, and departments of the Regular Army. Except as otherwise herein provided, a member of the Officers Reserve Corps shall not be subject to call for service in time of peace, and whenever called upon for service shall not, without his consent, be so called in a lower grade than that held by him in said reserve corps.

The President alone shall be authorized to appoint and commission as reserve officers in the various sections of the Officers Reserve Corps, in all grades up to and including that of major, such citizens as, upon examination prescribed by the President, shall be found physically, mentally and morally qualified to hold such commissions: *Provided*, That the proportion of officers in any section of the Officers Reserve Corps shall not exceed the proportion for the same grade in the corresponding arm, corps, or department of the Regular Army, except that the number commissioned in the lowest authorized grade in any section of the Officers Reserve Corps shall not be limited.

* * * * * * * *

The commissions of all officers of the Officers Reserve Corps shall be in force for a period of five years unless sooner terminated in the discretion of the President. Such officers may be recommissioned, either in the same or higher grades, for successive periods of five years, subject to such examinations and qualifications as the President may prescribe and to the age limits prescribed herein: *Provided*, That officers of the Officers Reserve Corps shall have rank therein in the various sections of said Reserve Corps according to grades and to length of service in their grades.

Sec. 38.—The Officers Reserve Corps in War.—In time of actual or threatened hostilities the President may order officers of the Officers Reserve Corps, subject to such subsequent physical examinations as he may prescribe, to temporary duty with the Regular Army in grades thereof which can not, for the time being, be filled by promotion, or as officers in volunteer or other organizations that may be authorized by law, or as officers at recruit rendezvous and depots, or on such other duty as the President may prescribe. While such reserve officers are on such service they shall, by virtue of their commissions as reserve officers, exercise command appropriate to their grade and rank in the organizations to which they may be assigned, and shall be entitled to the pay and allowances of the corresponding grades in the Regular Army, with increase of pay for length of active service, as allowed by law for officers of the Regular Army, from the date upon which they shall be required by the terms of their orders to obey the same: *Provided*, That officers so ordered to active service shall take temporary rank among themselves and in their grades in the organizations to which assigned, according to the dates of orders placing them on active service; and they may be promoted, in accordance with such rank, to vacancies in volunteer organizations or to temporary vacancies in the Regular Army thereafter occurring in the organizations in which they shall be serving. * * *

Any officer who, while holding a commission in the Officers Reserve Corps, shall be ordered to active service by the Secretary of War shall, from the time he shall be required by the terms of his order to obey the same, be subject to the laws and regulations for the government of the Army of the United States, in so far as they are applicable to officers whose permanent retention in the military service is not contemplated.

Sec. 39.—Instruction of Officers of the Officers Reserve Corps.—To the extent provided for from time to time by appropriations for this specific purpose, the Secretary of War is authorized to order reserve officers to duty with troops or at field exercises, or for instruction, for periods not to exceed fifteen days in any one calendar year, and while so serving such officers shall receive the pay and allowances of their respective grades in the Regular Army: *Provided*, That, with the consent of the reserve officers concerned, and within the limit of funds available for the purpose, such periods of duty may be extended for reserve officers as the Secretary of War may direct. * * *

* * * * * * * *

Sec. 55.—The Enlisted Reserve Corps.—For the purpose of securing an additional reserve of enlisted men for military service with the Engineer * * * Corps * * * of the Regular Army, an Enlisted Reserve Corps, to consist of such number of enlisted men of such grade or grades as may be designated by the President from time to time, is hereby authorized, such authorization to be effective on and after the first day of July, nineteen hundred and sixteen.

There may be enlisted in the grade or grades hereinbefore specified, for a period of four years, under such rules as may be prescribed by the President, citizens of the United States, or persons who have declared their intentions to become citizens of the United States, subject to such physical, educational and practical examination as may be prescribed in said rules. For men enlisting in said grade or grades certificates of enlistment in the Enlisted Reserve Corps shall be issued by the Adjutant General of the Army, but no such man shall be enlisted in said corps unless he shall be found physically, mentally and morally qualified to hold such certificate and unless he shall be between the ages of eighteen and forty-five years. The certificates so given shall confer upon the holders when called into active service or for purposes of instruction and training, and during the period of such active service, instruction, or training, all the authority, rights and privileges of like grades of the Regular Army. * * * And the Secretary of War is hereby authorized to issue to members of the Enlisted Reserve Corps and to persons who have participated in at least one encampment for the military instruction of citizens, conducted under the auspices of the War Department, distinctive rosettes or knots designed for wear with civilian clothing.

The President is authorized to assign members of the Enlisted Reserve Corps as reserves to particular organizations of the Regular Army, or to organize the Enlisted Reserve Corps, or any part thereof, into units of detachments of any arm, corps, or department in such manner as he may prescribe, and to assign to such units and detachments officers of the Regular Army or of the Officers Reserve Corps, herein provided for.

To the extent provided from time to time by appropriations, the Secretary of War may order enlisted men of the Enlisted Reserve Corps to active service for purposes of instruction or training for periods not to exceed fifteen days in any one calendar year: *Provided*, That, with the consent of such enlisted men and within the limits of the funds available for such purposes, such periods of active service may be extended for such number of enlisted men as may be deemed necessary.

Enlisted men of the Enlisted Reserve Corps shall receive the pay and allowances of their respective grades, but only when ordered into active service, including the time required for actual travel from their homes to the places to which ordered and return to their homes. * * *

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period June 10, 1916 to July 10, 1916:

Arthur Adams, Miami, Ariz.
J. H. Batcheller, Mattapoisitt, Mass.
R. S. Bonsib, Low Moor, Va.
E. L. Booth, New York, N. Y.
C. L. Bryden, Scranton, Pa.
F. Crabtree, Pittsburgh, Pa.
L. B. Eames, Pueblo, Colo.
S. J. Gormly, Guayacan, Chile.
Ernest A. Hersam, Berkeley, Cal.
A. W. Hudson, Douglas, Ariz.
Chas. W. Johnston, West Norfolk, Va.
John M. Lovejoy, New Rochelle, N. Y.
J. F. Manning, Seoul, Korea.
E. P. Mathewson, Anaconda, Mont.
Edwin W. Mills, Seoul, Korea.
Arthur Notman, Bisbee, Ariz.
L. B. Reifsneider, Daiquiri, Cuba.
W. S. Schuyler, Berkeley, Cal.
O. N. Scott, Toronto, Ont., Canada.
Arthur F. Taggart, New Haven, Conn.
Warren D. Thompson, Boston, Mass.
Thor. Warner, Kingman, Ariz.

Rodolphe L. Agassiz has been chosen president of the Calumet & Hecla Mining Co., an office which has been vacant for over a year.

Percy E. Barbour, Managing Editor of the *Engineering and Mining Journal*, resigned his commission as 2d Lieutenant in the Main Coast

Artillery to accept a commission as 1st lieutenant in the 22d Corps of Engineers, N. G. N. Y. He is now at Fort McAllen, Tex.

H. Kenyon Burch, Chief Engineer of the Inspiration Consolidated Copper Co., completed his work pertaining to the design and construction of the plant, and left on July 1st for an extended vacation trip throughout the East. His forwarding address for the next few months will be care of The Sierra Madre Club, L. A. Investment Bldg., Los Angeles, Cal.

Corey C. Brayton and **E. R. Richards** have established offices in the Hobart Bldg., San Francisco, Cal., for the purpose of conducting the business of Mining and Metallurgical Engineering.

Juan Felix Brandés has removed his offices to 139 Whittell Bldg., 166 Geary St., San Francisco, Cal.

S. A. Crandall has accepted a position as draftsman with the Bunker Hill and Sullivan Smelter, Kellogg, Ida.

John C. Devine, since 1906 assistant superintendent of the Ray Consolidated Copper Co., has resigned and will devote his attention to the development of the Renfro property of the Pinal Development Co., of which he is president.

Edward B. Durham is now with the Mammoth Copper Mining Co., Kennett, Cal.

Edward L. Estabrook has become geologist for the Wisconsin Zinc Co., at Platteville., Wis.

F. Julius Fohs, who has devoted himself to a consulting practice in oil geology for the past four years, announces that he has opened new offices in the Gallais Building at Tulsa, Okla., with a staff of six geologists and one topographer, and that he is covering an increasing scope of country with his structural surveys.

George A. Guess, professor of metallurgy at the University of Toronto, has been engaged by the Vermont Copper Co. to start up the copper smeltery at South Strafford, Orange Co., Vt., which has never been operated successfully.

Henry M. Howe has just been appointed Honorary Vice-President of the Iron and Steel Institute of Great Britain. There are only two others, Professor D. Tschernoff, of Russia; and Giorgio E. Flack, of Italy.

Capt. Robert W. Hunt has been honored by the Rensselaer Polytechnic Institute, with which he has been long identified, with the degree of Doctor of Engineering.

E. T. Lednum has been made manager of E. I. DuPont de Nemours & Co., with offices in Suite 808, Frisco Bldg., Joplin, Mo.

E. Fleming L'Engle has been appointed manager of the Royal Zinc Co., Joplin, Mo.

Waldemar Lindgren, Professor of Economic Geology at Massachusetts Institute of Technology, has been awarded the honorary degree of Doctor of Science by Princeton University.

N. O. Lawton has completed the exploration work in which he was engaged in Hancock Co., Tenn., and will be at South Strafford, Vermont, during July, examining copper properties.

C. H. Macnutt has resigned his position as manager of Burma Mines Co., Namtu, Burma.

W. G. Matteson has accepted the position of geologist for the Empire Gas and Fuel Co., Bartlesville, Okla.

William B. Phillips has resigned the presidency of the Colorado School of Mines, and has returned to Austin, Texas, to resume practice as a mining engineer and metallurgist.

Fred S. Porter has accepted a position in the engineering office of the Canadian Klondike Mining Co., Dawson, Yukon Territory, Canada.

F. Danvers Power has opened new offices in the Perpetual Trustee Chambers, Hunter St., Sydney, N. S. W., Australia.

H. W. Ross has accepted the position of Assistant Manager of the Backus & Johnston Co., Casapalca, Peru.

J. D. Sperr has resigned as mining engineer of the Tom Reed Gold Mines Co., Oatman, Ariz.

F. W. Traphagen has resigned the position of professor of metallurgy of the Colorado School of Mines to accept the presidency of the Colorado Metal Mining & Reduction Co.

Carl J. Trauerman, mining engineer, has resigned his position as mill superintendent of the August Mining Co., Landusky, Mont., and is inspecting the properties of the Beaver Creek Mines Co., Zortman, Mont. After examining properties in the Kendall, Elliston and York districts of Montana he will return to his headquarters at Butte.

Jesse Merrick Smith and **Miss Mabel Amy MacKinney** were married on June 24, at Tarrytown-on-the-Hudson, N. Y.

Edward Vestal Graybeal and **Miss Alice Seabury Mathewson** were married on June 14, at Anaconda, Mont.

POSITIONS VACANT

Wanted, for a new smelting company now being organized in China: one man familiar with the design and operation of a zinc smelter; one experienced in the design and operation of a lead smelter; and a chemist competent to make the usual analyses in connection with lead and zinc smelting and to make clay analyses. Men with their wives preferred. Company will pay all expenses to and from China, medical attendance, salary, bonus per ton of zinc or lead produced, allowance for food, and furnish quarters. No. 120.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, oil geologist, age 33 years, technical education, nine years' experience with oil and mining companies, wishes employment with reputable company or syndicate exploiting proven oil lands or searching for new districts. First class references. No. 300.

Member, age 30, unmarried, Columbia graduate, speaks fair French, and writes and speaks Spanish fluently. Mining and geological experience in United States, Mexico and South America. Desires position with examining engineers or exploration company. No. 301.

Member, technical graduate in mining engineering, three years' experience in surveying, sampling and underground engineering in metal mines of the West, desires position as mine surveyor, field engineer or assistant to consulting engineer. No. 302.

Member, age 30, technical graduate, 4 years' experience in tin concentration; at present Superintendent of Mills in largest Bolivian tin mines; executive ability; seeks change, preferably with American company interested in locating or investigating in South America. Fluent Spanish. No. 303.

Open for engagement. Master Mechanic. A man experienced in smelter construction and operation. Lead and copper especially. 18 years as M. M. in British Columbia and Mexico. Reliable, efficient, with good record. Best of reference. No. 304.

Engineer, 42 years old, married, experienced in cyanidation, concentration and construction of plants, desires position as operator or construction engineer or in plant designing. Speaks Spanish and some German. No. 305.

Mining engineer with 6 years' operating experience in Colorado and Arizona desires position with substantial company. Employed at present. Age, 30. Married. References. No. 306.

Member, technical graduate, who has specialized in the treatment of gold and silver ores, desires position as mill superintendent or foreman; 7 years' practical experience. Highest references as to character and ability to get results. No. 307.

Member with broad general business experience. Past 5 years president and general manager of active developing and producing oil company in California. Open for engagement in California preferred. No. 308.

Member, technical metallurgical graduate. 5 years' experience in various mining operations. Mill and laboratory work including concentrating, cyaniding and flotation. Desires position in flotation work. Age, 30. Married. No. 309.

Member, aged 30, technical graduate in mining and metallurgy. At present employed as assistant superintendent of concentrator and flotation plant. Also had practical and engineering mining experience. Desires change. No. 310.

Member, aged 26, graduate in mining and metallurgy, 2 years' experience with engineering work of lead and copper smelters, good draftsman; desires permanent connection with engineering or operating department of smelter. Excellent references. At present employed, but available at short notice. No. 311.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

Apparatus for Making Photographic Reproductions

The United Engineering Society has installed a photographic duplicating machine for the reproduction of pages from books and periodicals, maps and charts in the Library of the Engineering Societies, and it is now in operation. Orders may be sent to the Library of the Engineering Societies, 29 West 39th Street, New York.

The process produces a photograph (white on black) direct on bromide paper, and from this negative print a positive may be made by re-photographing. The charge for a print 11 by 14 in., which may cover two pages of an octavo volme that face each other, is 25 cents for a negative print and 50 cents for a positive print. Enlargements and reductions to 11 by 14 in. are easily made. The process has the advantages of accuracy and cheapness.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

BY-PRODUCTS RECOVERED IN THE MANUFACTURE OF COKE. By W. H. CHILDS. n.p., 1916. (Gift of D. D. Berolzheimer.)

COAL DISTILLATION, GASIFICATION AND BY-PRODUCTS. By J. E. CHRISTOPHER WIGAN, 1915.

THE ANALYSIS OF PERMISSIBLE EXPLOSIVES. Bull. 96, U. S. Bureau of Mines. Washington, 1916.

SAND TEST FOR DETERMINING THE STRENGTH OF DETONATORS. Tech. Paper No. 125, U. S. Bureau of Mines. Washington, 1916.

ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, REPORTED FROM OCTOBER TO DECEMBER, 1915. Bull. No. 118, U. S. Bureau of Mines. Washington, 1916.

EFFECTS OF ATMOSPHERES DEFICIENT IN OXYGEN ON SMALL ANIMALS AND ON MEN. Tech. Paper No. 122, U. S. Bureau of Mines. Washington, 1915.

MINERS' NYSTAGMUS. Bull. No. 93, U. S. Bureau of Mines, Washington, 1916.

LIST OF BOOKS RELATING TO THE IRON AND STEEL INDUSTRY. Contained in the principal libraries in the West of Scotland, Glasgow, 1916. (Gift of West of Scotland Iron and Steel Institute.)

Geology and Mineral Resources

REVIEW OF PRINCIPLES OF OIL AND GAS PRODUCTION. Compiled by ROSWELL H. JOHNSON and L. G. HUNTLEY. John Wiley & Sons, Inc., New York, 1916. Price $3.75. Gift of publishers.

[A book on oil by the two distinguished authors of this volume is sure to arouse great interest among members of the Institute who are familiar with the writings of these same authors in our publications. To such it is not necessary for us to say anything as to the accuracy, care, and thoroughness of the work, and we commend the volume to all. It is well and appropriately illustrated by geological sketches, drawings, curves, photographs, and so forth, and it covers the following subjects:

Varieties of Oil and Gas, The Origin of Oil and Gas, Distribution of the Oil and Gas, Reservoirs of Oil and Gas, Accumulation of Oil and Gas, Pressure in Oil and Gas Reservoirs, Origin of the Shape of the Reservoir, Classification of the Attitude of Geologic Surfaces, Effect of the Different Attitudes upon Accumulation, Locating Oil and Gas Wells, Oil and Gas Lands, Drilling for Oil and Gas, Bringing in a Well, The Management of Oil Wells, Completing the Extraction of the Oil, The Management of Gas Wells, Condensation of Gasoline from Gas, The Natural Gas Industry, Size and Scope of Oil and Gas Companies, Reports upon Oil and Gas Prospects or Properties, The Valuation of Oil Properties, Oil and Gas Fields of North America, Oil Market and the Future Supply.]

DEPOSITS OF THE USEFUL MINERALS AND ROCKS, THEIR ORIGIN, FORM AND CONTENT. By F. Beyschlag and others. Vol. II, London, 1916.

OIL AND GAS MAP OF SOUTHWESTERN PENNSYLVANIA, 1915. Harrisburg, 1916.

THE CADDO OIL AND GAS FIELD, LOUISIANA AND TEXAS. Bull. No. 619, U. S. Geological Survey. Washington, 1916.

PETROLEUM AND NATURAL GAS PROSPECTS AT ROMA. Publication 247, Queensland Geological Survey. Brisbane, 1915.

TUNGSTEN AND MOLYBDENUM. Part I–II. (Tasmania. Geological Survey Mineral Resources, No. 1.) Tasmania, 1916.

THE WOLFRAM, MOLYBDENITE AND BISMUTH MINES OF BAMFORD, NORTH QUEENSLAND. Publication 248, Queensland Geological Survey. Brisbane, 1915.

FELDSPARS OF THE NEW ENGLAND AND NORTH APPALACHIAN STATES. Bull. No. 92, U. S. Bureau of Mines. Washington, 1916.

ZINC-LEAD SULPHIDE DEPOSITS OF THE READ-ROSEBERY DISTRICT PT. II.—Rosebery group with maps. Bull. No. 23, Tasmania Geological Survey. Tasmania, 1915.

ATLANTIC GOLD DISTRICT AND THE NORTH LARAMIE MOUNTAINS, FREMONT, CONVERSE, AND ALBANY COUNTIES, WYOMING. Bull. No. 626, U. S. Geological Survey. Washington, 1916.

GEOLOGY AND COAL RESOURCES OF CASTLE VALLEY IN CARBON, EMERY, AND SEVIER COUNTIES, UTAH. Bull. 628, U. S. Geological Survey. Washington, 1916.

GEOLOGY AND MINERAL RESOURCES OF THE SOUTHERN COAL FIELD. Part I—SOUTH COSTAL PORTION., With Maps and Sections. New South Wales. Memoirs of the Geological Survey. Geology No. 7. Sydney, 1915.
GEOLOGY AND ORE DEPOSITS OF ROSSLAND, BRITISH COLUMBIA. Memoir No. 77. Canada Dept. of Mines. Ottawa, 1915.
GEOLOGY OF FIELD MAP-AREA, B. C. AND ALBERTA. Memoir No. 55, Canada Dept. of Mines. Ottawa, 1914.

General

CIVIL ENGINEERS' COST BOOK. Ed. 2. By T. E. Coleman. London, 1916.
DIRECTORY OF THE IRON AND STEEL WORKS OF THE UNITED STATES AND CANADA, 1916. 18th Ed. New York, 1916.
LAND AND MARINE DIESEL ENGINES. By Giorgio Supino. Translated by A. G. Bremner and James Richardson. London, 1915.
THE DIESEL ENGINE IN PRACTICE. By J. E. Megson and H. S. Jones. San Francisco, 1916.

Gift of Hill Publishing Company

A good collection of 29 text books.

Trade Catalogs

E. I. DUPONT DE NEMOURS & CO. Wilmington, Del. DuPont Products (A-430). April, 1916.
SULLIVAN MACHINERY CO., Chicago, Ill. Bull. 70-C. The Sullivan "Dr-6" mounted water hammer drill. May, 1916.
SPARTA IRON WORKS CO., Sparta, Wis. Sludge Bucket. June, 1916.
STEPHENS-ADAMSON MFG. CO., Aurora, Ill. Labor Saver. June, 1916.
TEXAS CO., Houston & New York. About Texaco crater compound, 1916.

Exposition of Chemical Industries

That the chemical industries are alive to the situation that they now occupy in the sum of affairs is evident in the preparations being made by many firms for the Second National Exposition of Chemical Industries to be held at the Grand Central Palace, New York City, during the week of September 25–30, 1916. The list of exhibitors indicates that the Exposition is already twice the size of its successful predecessor.

The American Chemical Society will hold its annual meeting during the whole week, the American Electrochemical Society has arranged to hold its meetings on September 28, 29 and 30, and the Technical Association of the American Pulp and Paper Industry is also planning to meet during the week.

The Bureau of Commercial Economics at Washington is again coöperating with the Exposition by arranging an elaborate program of motion pictures covering subjects dealing with the industries depending on chemistry. Many of these films will have their first showing at the Exposition. They will take the visitor through industrial plants, shops, manufactories, mines, etc., where visitors are never admitted; they will show operations that in ordinary light would never be seen; they will demonstrate more than any other kind of motion picture the great strides and improvement in the art of photography.

MEMBERSHIP

New Members

The following list comprises the names of those persons who became members during the period June 10, 1916, to July 10, 1916:

Adams, William S., Filtration Engr., Anaconda Copper Min. Co., 701 Cherry, Anaconda, Mont.
Alley, James E., Shift-boss............Belmont Milling Co., Tonopah, Nev.
Badgley, Charles Welling, Research Chemist, El Paso Smelting Works, El Paso, Tex.
Ballenberg, Adolf G., Engr., Porcupine Exploration Co., Ontonagon Co., Mich.
Barker, Ralph Feiner, Met..................Garfield Smelt. Co., Garfield, Utah.
Blossom, Edward Ludlam, Staff of Research Corpn., 63 Wall St., New York, N. Y.
Broughton, Eugene Harding, Experimental Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.
Burford, Samuel W., Genl. Mgr., Cleveland Mining Co., 223 West Third St., Joplin, Mo.
Butterfield, George Bruce, Inspector, The Associated Companies, First National Bank Bldg., Pittsburgh, Pa.
Cahill, Edward Gustav, Contracting Engr., 460 Montgomery St., San Francisco, Cal.
Daulton, Theodore M....................208 New York Bldg., Seattle, Wash.
Deane, Elbridge Gerry, Min. Engr..........Miami Copper Co., Miami, Ariz.
Duncan, L. Grant, Met. Engr.................Mineral Point Zinc Co., Depue, Ill.
Eisele, George J., Asst. Genl. Supt., Oliver Iron Mining Co., Iron Mountain, Mich.
Flynn, John G.....................Mining, Miami Copper Co., Miami, Ariz.
French, R. W., Met...............Goldfield Cons. Mines Co., Goldfield, Nev.
Freud, Benjamin Ball, Teacher, Associate Prof. of Chemistry, Armour Institute of Technology, Chicago, Ill.
Garlock, Guy M., Fuel Dept., Prime Western Spelter Co., Box 83, Iola, Kans.
Geohegan, Leslie E.......Asst. to Prest., Gulf States Steel Co., Birmingham, Ala.
Goodale, F. A., Supt. and Agent, You Bet Mining Co., P. O. Box. 290, Dutch Flat, Cal.
Handy, Royal Sheppart, Mill Supt., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
Harding, Philip Orr, Engr., Designing Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
Hensley, James H. Jr., Min. Engr., Miami Copper Co., Box 100, Miami, Ariz.
Hershey, Oscar H., Min. Geol., Burch, Caetani & Hershey, Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
Hoffmann, Robert Bernard, Chief of Service Dept., Mineral Point Zinc Co., Depue, Ill.
Hull, Monroe Richmond.....Chief Engr., Arizona Copper Co., Clifton, Ariz.
Jones, J. Claude, Prof. of Geol............University of Nevada, Reno, Nev.
Kelsey, W. M., Genl. Supt................Mineral Point Zinc Co., Depue, Ill.
Liddell, Donald M., Cons. Met. Engr.. Merrill, Lynch & Co., 7 Wall St., New York, N. Y.
Lunt, Charles Frederick, Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
Mann, Horace Tharp, Associate Prof. of Met. and Ore Dressing, Missouri School of Mines and Metallurgy, P. O. Box 248, Rolla, Mo.
Mendelsohn, Charles, Mech. Engr., Old Dominion Copper Min. & Smelt. Co,. Box 657, Globe, Ariz.
Moore, George Pierson, Engr., The Henry Souther Engrg. Co., 11 Laurel St., Hartford, Conn.
Read, Norman Hatfield, Min. Geol.......................Manchester, Mass.
Robbins, O. D., Met. Engr., Chief of Ore and Testing Dept., Mineral Point Zinc Co., Depue, Ill.
Scott, David Burnet, Min. Engr., Efficiency Engr., Miami Copper Co., Box 100, Miami, Ariz.
Scott, J. Ralph, Mine Engr., Dome Mines Co., Ltd., Lock Box 505, So. Porcupine, Ont., Canada.
Smith, W. R., Supt..................Mineral Point Zinc Co., Mineral Point, Wis.

STAY, THERON D., Asst. Research Met., Aluminum Castings Co.,
2077 East 88th St., Cleveland, O.
STEIDLE, EDWARD, Technical Asst. to Chief Min. Engr., U. S. Bureau of Mines,
Washington, D C.
STOCKETT, NORMAN AYLEWORTH, Min. Engr., Mgr., Paragon Cons. Min. Co.,
Paragon, Idaho.
STRACHAN, CHARLES B., Mill Supt., American Zinc Co. of Tennessee, Mascot, Tenn.
STRAUB, CHARLES EDWARD, Cons. Oil Geol. & Engr.,
211-12 Lynch Bldg., Tulsa, Okla.
STROUP, THOMAS A., Mech. and Met. Engr., Tennessee Copper Co., Copperhill, Tenn.
TODD, W. PARSONS, Vice-Prest., Quincy Min. Co., 32 Broadway, New York, N. Y.
WATSON, HUGH C., Min. Engr., Genl. Supt., Wisconsin Zinc Co., Platteville, Wis.
WARD, ARTHUR THOMAS, Experimental Engr., Braden Copper Co., Sewell,
via Rancagua, Chile, S. Amer.
WEINTZ, LOUIS JOHN, Mill Supt., The New Jersey Zinc Co.,
Ogdensburg, Sussex Co., N. J.
WHITE, HENRY ARTHUR, Cons. Met., Geduld Prop. Mines, Ltd., Box 41,
Springs, Transvaal, S. Africa.
WOODWARD, TRUMAN STEPHEN, Chief Chemist, Ohio Works, Carnegie Steel Co.,
R. 2. Youngstown, O.
YERXA, R. B., Asst. Supt. of Concentrator, Miami Copper Co.,
P. O. Box 100, Miami, Ariz.
YOUNG, EARL BURDETT, Asst. in Geol. Dept., Anaconda Copper Min. Co.,
526 Hennessy Bldg., Butte, Mont.

Associate Members

CAJANDER, A. W., Draftsman, Braden Copper Co., Rancagua, Chile, S. Amer.
RIVES, HENRY MACON, Secy. and Treas., Nevada Mine Operators Assn.,
210-211 Reno National Bank Bldg., Reno, Nev.
ROBINSON, CHARLES H., Iron Mining, Pettit & Robinson Lands,
826 First National Soo Line Bldg., Minneapolis, Minn.

Junior Members

BERTHIER, ULYSSES HENRY, Asst. in Testing Dept., International Smelt. Co.,
Box 115, Tooele, Utah.
BIRKETT, HOWARD, Student..........Columbia University, New York, N. Y.
BLAUROCK, CARL ALBERT, Gold and Silver Refiner, 2530 W. 37th Ave., Denver, Colo.
CHATIN, AUGUST H., Laborer, Engr. of Mines, Anaconda Copper Co.,
316½ W. Copper St., Butte, Mont.
CORBY, HARRY GILBERT, Student................1138 S. Maple St., Carthage, Mo.
CRAIG, JOHN J., Min. Engr., Northwestern Improvement Co., Brainerd, Minn.
EHLERS, LOUIS WILMER, Student..........Missouri School of Mines, Rolla, Mo.
ERDOFY, MAXWELL EMORY, Student.....Columbia University, New York, N. Y.
FERGUSON, KENNETH SEARS, Student, Colorado School of Mines, Golden, Colo.
FLOSS, FRITZ CARL, Student..................................Milwaukie, Ore.
HOFIUS, MAX T., Student..........Colorado School of Mines, Golden, Colo.
HU, SHIH-HUNG, Student.....................P. O. Box 732, Golden, Colo.
KLEIN, KARL F., Min. Engr., Morococha Mining Co., Morococha, Peru, S. Amer.
PIERCE, COLWELL ARBA, Student......3417 Virginia Ave., Kansas City, Mo.
SCHMIDT, ROBERT DROEGE, Student..........1509 Alameda Ave., Lakewood, O.
STOTZ, NORMAN I., Student...............Lehigh University, S. Bethlehem, Pa.
SUTTON, HARRY ALLEN, Senior Student, Oregon Agricultural College, Corvallis, Ore.
WIEBELT, FRANK J., Min. Engr., Laborer, Anaconda Copper Co.,
217 Colorado St., Butte, Mont.

Total Membership, July 10, 1916........................ 5,612

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period June 10, 1916 to July 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Robert Sexton Allen, Globe, Ariz.
Proposed by Rudolf Gahl, Guy H. Ruggles, C. E. Arnold, W. B. Cramer.
Born 1889, New Haven, Conn. 1896–1906, Grade and High School, Kansas City, Mo. 1906–09, Missouri School of Mines. 1909–12, Lab. Work and Control Chem., Nev. Cons. Copper Co. 1912–13, Slag Chemist, Old Dominion Copper Co. 1913–14, Chem. and Asst. Engr., Superior & Boston Copper Co. 1914–16, Sr. Control Chem., Nev. Cons. Copper Co.
Present position: Chief Chem. of Concentrator, Inspiration Cons. Copper Co.

Frederick Eurith Beasley, Tacoma, Wash.
Proposed by Willard V. Morse, Roger E. Chase, Brent N. Rickard.
Born 1882, Peoria, Ill. 1891–1901, Elementary and High School, Peoria, Ill. 1901–04, Mech. Engrg., Univ. of Illinois, Urbana, Ill. 1902, Experimental Dept., Acme Harvest Machine Co., Peoria, Ill. 1903, Civil Engrg. Dept., Toledo, Peoria & Western R. R. 1904–05, Asst. Engr., Canadian Min. & Smelt. Co., Chem. Dept., Trail Smelter, Trail, B. C. 1905–16, Representative Engr., Tacoma Smelter for Trail Smelter shipments. 1910–12, Asst. Representative, Treadwell Min. Co., Tacoma, Wash. 1914, Asst. Representative, Kennicott Min. Co., Tacoma, Wash. 1912–13, Asst. to A. McCullough, Met., Tacoma, Wash.
Present position: Resident Engr., Consolidated Min. & Smelt. Co., Tacoma, Wash.

Howard A. Beck, Aurora, Nev.
Proposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
Born 1892, New York, N. Y. Grammar and High School, New York City and Denver. 1912–13, Mine Foreman, Leadville Mines Co., Lead, Mo. 1913–14, Night Foreman, St. Kevin Min. Co., Lead, Mo. 1915, Aurora Cons. Mine and Mill. 1915, Shift Foreman, Golden Boulder Co., Fairview, Nev. 1915 to date, Filterman, Tank man, Tube mill man, Aurora Cons. Mines Co.
Present position: Shift Foreman, Aurora Cons. Mines Co. Mill.

P. McDonald Biddison, Columbus, O.
Proposed by Samuel S. Wyer, Harry C. Reeser, Daniel M. Armstead.
Born 1884, Lane, Kan. 1904, Kansas State Agricultural College, B. Sc. in Elec. Engrg. 1904, Draftsman and Electrician, Cement Plant Const., Hunt Engrg. Co., Iola, Kansas. 1905, Sta. Engr., Iola Portland Cement Co., Iola, Kans. 1905, Electrician, Zeigler Coal Co., Zeigler, Ill. 1905, Draftsman. 1905–06, Chief Engr., Logan Nat. Gas & Fuel Co., Columbus, O. 1906, Sales Engr., Hope Engrg. & Supply Co., Joplin, Mo. 1907, Asst. Chief Engr., Kansas Nat. Gas Co., Independence, Kans. 1908, Const. Engr., Corcoran Cons. Co., Huntington, W. Va. 1909, Const. Engr., Columbia Gas & Electric Co., Kenova, W. Va. 1910, Mech. Engr., Ohio Fuel Supply Co., Homer, O. 1911, Mech. Engr., The Texas Co., Port Arthur, Tex. 1912–16, Const. Engr., The Ohio Fuel Supply Co., Columbus, O.
Present position: Const. Engr., The Ohio Fuel Supply Co.

Paul Truman Boise, Salt Lake City, Utah.
Proposed by Charles P. Brooks, Charles W. Stimpson, J. C. Jones.
Born 1888, Lincoln, Kans. 1904–06, Univ. of Utah. 1906–07, Michigan College of Mines. 1908, Shift Boss, Gemini Min. Co., Eureka, Utah. 1909–10, Supt., Yosemite Mines Co., Bingham, Utah. 1911–14, Shift Boss, Silver King Coalition Mines Co., Park City, Utah.
Present position: 1914 to date; Stimpson Equipment Co.

Ralph H. Bourne, Chicago, Ill.
Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.
Born 1881, Cleveland, O. 1905, Grad., Cornell Univ., Mech. Engrg. Member of Amer. Soc. of Mech. Engineers. 1906, Draftsman, 8 months, Wellman Seaver-Morgan Co. 1907, 8 months, Hoover & Mason, Chicago, Ill.
Present position: Vice-Pres. & Asst. Sales Mgr., Whiting Foundry Equipment Co., Harvey, Ill.

Henri Brings, Paris, France.
Proposed by E. Gybbon Spilsbury, J. E. Kennedy, P. De P. Ricketts.
Born 1879, Duren. 1889–97, Grad., Realgymnasium, Aix-la-Chapelle. 1898–1902, Technical High School, Aix-la-Chapelle. 1902, Grad. Min. & Met. Engr., Technical High School, Aix-la-Chapelle. 1902, Min. Engr. Ivory Coast Exploring Syndicate, Ltd., Salisbury House, London. Exploration work in French West Africa. 1903, Surveying and exploration work, French Guinea and Senegal for same company. 1904–07, Chief Engr. and Genl. Mgr., Société des Dragages Auriferes du Tinkisso, Paris, France. 1908–10, Chief Engr., Cie. Minière de l'Ouest Africain, Paris, France.
Present position: 1910 to date; Chief Engr. & Mgr., Cie. Formiere des Mines d'Or d'El Dorado, Paris, France.

Percy Bennett Butler, Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1876, San Francisco, Cal. 1893, Grad., Commercial High School, San Francisco, Cal. 1913, Student, International Correspondence School, mining course. 1899, Parral Mill Co., Parral, Chih., Mex. 1900, Cia. Restauradora, Guanacevi, Mex. 1903, Montezuma Lead Co., Santa Barbara, Chih., Mex. 1905, U. S. Min. Co., Minas Nuevas, Chih., Mex. 1907, Veta Colorada Min. & Smelt. Co., Minas Nuevas, Chih., Mex. 1907–12, Operating for own account in Parral and Santa Barbara, Mex.
Present position: 1914 to date; Mgr. of mines in Joplin district.

Robert Stuart Butler, Joplin, Mo.
Proposed by A. F. Truex, G. B. Corless, C. A. Wright.
Born 1891, San Francisco, Cal. 1907, Berkeley High School, Berkeley, Cal. Student, International Correspondence Schools. 1910–13, Operating in Santa Barbara and Parral, Mex. 1914, Supt. of Paragon and other properties in Joplin district.
Present position: Supt.

Allan B. Calhoun, Globe, Ariz.
Proposed by W. B. Cramer, J. C. Low, E. M. Marshall.
Born 1879, Greenock, Scotland. Grade and high school education in public schools, Minneapolis, Minn. 1905, Grad., Minnesota School of Mines, E. M. 1906–07, Assayer, Shift boss, Foreman, Cornucopia Mines of Oregon, Cornucopia, Ore. 1908–09, and 1911, Engr., Cons. Min. & Smelt. Co. of Canada, Rossland, B. C. 1912–13, Supt., Motherlode Sheep Creek Min. Co., Sheep Creek, B. C. 1914, Supt., Arabian Cons. Mines, Kingman, Ariz. 1905, Supt., El Rayo Mine, Santa Barbara, Mex.
Present position: Efficiency Engr., Old Dominion M. & S. Co.

Byron H. Carpenter, Benton, Ill.
Proposed by B. Schettler, Frank A. Ray, Oscar Cartlidge.
Born 1891, Mentor, O. 1905–09, Willoughby High School, Willoughby, O. 1910–14, Ohio State Univ., Columbus, O., E. M. 1909–10, Rodman, Clark & Pike, Topographers and Land Surveyors, Willoughby, O.
Present position: Min. Engr., The Benton Coal Co.

Allison Robert Chambers, New Glasgow, N. S.
Proposed by R. E. Chambers, Thomas Cantley, J. B. Porter.
Born 1879, Halifax, N. S. High Schools, New Glasgow & Halifax. Dalhousie Univ., Chemistry. 1900–04, McGill Univ., B. S. 1896–99, Chaining and Transit-Dan, Nova Scotia Steel Co., Nova Scotia and Newfoundland. 1899–1900, Civ. Engr., mominion Iron & Steel Co., Wabana, Newfoundland. 1904 to date, Min. Engr.,

Nova Scotia Steel & Coal Co., Ltd., Wabana. Resident Mgr., Wabana. Asst. Mgr., Ore Mines and Quarries, New Glasgow, N. S.

Present position: Asst. Mgr. on Ore Mines and Quarries.

Marcu F. Chase, St. Louis, Mo.

Proposed by George C. Stone, J. H. Janeway, G. S. Brooks.

Born 1876, Alton, Ill. 1897, Trinity College, Hartford, Conn., B. S. 1898–1900, Chem., Mineral Point Zinc Co. 1900–04, Acid Expert, New Jersey Zinc Co. 1904–12, Supt. of Depue Plant, Mineral Point Zinc Co. 1912–15, Genl. Mgr., Mineral Point Zinc.

Present position: Cons. Engr.

Gilbert Ernest Cheda, La Libertad, Nicaragua.

Proposed by E. W. Armstrong, Charles E. Brinker, S. M. Parker, F. B. Forbes.

Born 1884, San Luis Obispo, Cal. 1911, Dept. of Geol. and Min., Stanford Univ., Cal., A. B. 1905–06, Underground, Cananea Cons. Copper Co., Cananea, Mexico. 1908, Underground, Keystone Mine, Cal. 1909, Underground, Pittsburgh Silver Peak Mine, Nev. 1911–16, Engr. and Mine Foreman, Babilonia Gold Mines, Ltd., La Libertad, Nicaragua.

Present position: Mine Foreman, Babilonia Gold Mines.

Charles W. Clark, San Mateo, Cal.

Proposed by Horace V. Winchell, Robert E. Tally, W. A. Clark.

Born 1871, Deer Lodge, Mont.

Present position: Genl. Mgr., United Verde Copper Co., Jerome, Ariz.

Isaac N. Dally, Seattle, Wash.

Proposed by F. C. Greene, Henry L. Manley, Amos Slater.

Born 1872, Webster City, Iowa. 1896–1901, Private development work on metalliferous mines in British Columbia. 1901–05, Geological and Coal Surveys, British Columbia and Alberta. 1905–07, Halls Min. & Smelt. Co., Nelson, B. C.

Present position: 1909 to date; Coal Dept., Chicago, Milwaukee & St. Paul R. R.

W. Val De Camp, Mayer, Ariz.

Proposed by G. M. Colvocoresses, Robert E. Tally, J. L. White.

Born 1886, Cirsco, Iowa. 1908, Colorado School of Mines, E. M. 1908, Foreman, Atlas Min. Co., Ouray, Colo. 1909–13, Supt. and Mgr., Pacific Copper Co., Crown King, Ariz. 1913, Supt., Haynes Copper Co., Jerome, Ariz. 1914, Foreman and Boss, United Verde Copper Co., Jerome, Ariz. 1915–16, Supt., Blue Bell Mine, Mayer, Ariz.

Present position: Supt., Blue Bell Mine.

H. DeWitt Decker, Anaconda, Mont.

Proposed by Charles D. Demond, C. R. Wraith, Enoch A. Barnard.

Born 1892, La Porte, Ind. 1914, Grad. Wisconsin School of Mines. 1915, 3 mos., Machineman, Frontier Mining Co.. Benton, Wis.; 3 mos., Shaft and Machineman, Admiralty Zinc Co., Guapan, Okla. 1916, Washoe Reduction Works, Anaconda Copper Mining Co., Anaconda, Mont.

Present position: Testing Dept., Anaconda Copper Mining Co., Anaconda, Mont.

Axel Eric, Salt Lake City, Utah.

Proposed by R. C. Gemmell, H. C. Goodrich, D. C. Jackling.

Born 1879, Denmark. 1897, Grad., High School. 1897–1900, Machine Shop practice. 1907, Grad., Copenhagen Univ. of Polytechnic, Mech. Engr. 1906, Worked for Prof. Rung, projecting electrification of State Railroads near Copenhagen. 1907–09, Messrs. Lossen & Hjorth, London, England, steam and water softening plants. 1909, Franklin Railway Supply Co., N. Y. 1910, Midvale Steel Co. and American Bridge Co., Philadelphia. 1911, Robins Conveying Belt Co., N. Y. 1912–15, Designing and erecting Leaching Plant, Chile Exploration Co., N. Y., and Chile, S. A.

Present position: Designing Leaching Plant, Utah Copper Co.

C. L. French, Ely, Nev.

Proposed by J. A. Rule, A. M. Day, A. B. Young.

Born 1890, Charlotte, Mich. 1909, College Prep., Fort Collins, Colo. 1913, Colorado School of Mines, E. M. 1913, Classifier Man, Ray Cons. Copper Co., Hayden, Ariz.; Cyanide Millman and Assayer, Churchill Milling Co., Wonder, Nev. 1914–16, Cottrell Foreman, Testing Dept., Asst. Smoke Investigation Dept., Custom Ore Sampler, International Smelting Co., Tooele, Utah.

Present position: Millman, Independent Tungsten Co.

John Creighton Garvin, Spokane, Wash.
Proposed by L. K. Armstrong, J. McD. Porter, A. W. Paterson.
Born 1840, County Londonderry, Ireland. 1848–69, Common and Grammar Schools, 1873, Queens prize in Geology in Marghafalt National School, Dublin. 1873, Examinations Eastern States Mineral Belt, Colombia, S. Amer. 1881, 6 years gold and silver reduction works, Philadelphia. 1881–97, 10 years in Leadville; balance at Puzzler Mine, Boulder Co., in mining and metallurgical practice. 1897 to date, at Spokane in professional work.
Present position: Consulting Engineer.

Augustus Brunner Hardie, Philadelphia, Pa.
Proposed by A. W. Hudson, H. H. Colley, Utley Wedge.
Born 1886, Philadelphia, Pa. 1895–1903, Girard College, Philadelphia. 1903–04, Franklin Institute, Philadelphia. 1908, Univ. of Penn. 1910–12, Univ. of Wisconsin. 1903–05, Shop practice, etc., Queen & Co., Inc., Philadelphia. 1905–07, Draughtsman, Standard Steel Works, Pa. 1907–08, Draughtsman, General Electric Co., Philadelphia. 1909, Asst. Plant Engr., Stanley G. Glagg, Pa.; Die Designer, Pressed Steel Car Co., Pittsburgh. 1910–11, Const. Supt., Tennessee Copper Co., Tenn. 1912–14, Const. Supt., Braden Copper Co., Chile. 1914, Design, supervision of erection, metallurgical plants with Utley Wedge.
Present position: Supt. of erection, Sulphuric Acid Plant, for Utley Wedge at Calumet & Arizona Copper Co..

Fredrik Jörgen Ording Hurum, Pittsburgh, Pa.
Proposed by Stephen L. Goodale, Robert M. Black, Horatio C. Ray.
Born 1893, Christiania, Norway. 1911, Examination Artium, Christiania. 1911–14, Royal German Inst. of Tech., Aachen. 1914, Cand. rer. Met., Aachen, Germany. 1915–16, Univ. of Pittsburgh, Met. Engrg.
Present position: Met. Engr.; Univ. of Pittsburgh.

William Wallace Inglis, Scranton, Pa.
Proposed by H. M. Warren, J. M. Humphrey, Paul Sterling.
Born 1871, Scranton, Pa. High School, Scranton, Pa. Hillside Coal & Iron Co. Pennsylvania Coal, N. Y., Susquehanna & Western Coal Co.
Present position: Genl. Mgr., Coal Min. Dept., Delaware, Lackawanna & Western R. R. Co.

Junzo Inouye, Kenjiho, Koshin, Korea.
Proposed by Kenroku Ide, Shinji Harada, Takeshi Kawamura.
Born 1888, Osaka, Japan. 1913, Grad., Min. & Met. Institute, Imperial Univ., Kyoto, Japan. 1913, Mitsubishi Co., Tokyo, Japan.
Present position: Met. Engr., Iron & Steel Dept., Mitsubishi Co.

Phillips F. Jarvis, St. Louis, Mo.
Proposed by Frederick K. Copeland, P. N. Moore, E. J. Rossbach, O. M. Bilharz, H. T. Walsh.
Born 1878, Claremont, N. H. 1897, Stephens High School, Claremont, N. H. 1897, Sullivan Machinery Co., Chicago office. Shop practice, sale of mining machinery, metal mines and quarries, and specializing considerably in coal mining. Have been connected with metal mines and was president, Columbia Zinc Co., Joplin, Mo.
Present position: Mgr., Sullivan Machinery Co.

Edward Charles King, Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1875, Rockland, Mich. 1880–90, Public School. 1893, Bryant & Stratton Business College, Buffalo, N. Y. 1905, Special course in metallurgy, Michigan College of Mines. 1890–1900, Calumet & Hecla Smelteries, Lake Linden, Mich. & Buffalo, N. Y. 1900–04, In charge of the Casting Dept. and later Night Supt. Washoe Smeltery, Mont. 1904–05, Asst. Supt., Arizona Smelt. Co. 1905–07, Supt. of Smelter, Orogrande, N. M. 1907–10, Asst. Supt. of Smelter, Cananea Cons. Copper Co., Cananea, Mex. 1910–13, Met. on staff of southern division, Amer. Smelt. & Ref. Co.
Present position: 1914 to date; Supt. of Smelters, Cons. Arizona Smelt. Co.

Victor E. Lieb, Elephant Butte, N. Mex.
Proposed by L. J. Charles, S. H. Worrell, William B. Phillips.
Born 1883, Brenham, Tex. 1900–02, Academic course, Polytechnic Inst. of Brooklyn. 1903–07, Min. Engrg., Univ. of Tex., worked for W. Waldo and F. A. Jones of Houston, Tex., general engineering practice. 1907–08, Supt., Las Machos

Min. Co., Tude Durango, Mex. 1908–09, Mine examination, prospecting and development of own property, Chih., Mex.; chem. and engr., Southern Min. Co. Ocampo, Chile. 1909–10, Engr., Candelaria Min. Co., San Pedro, Chih., Mex. 1910 to present, Engr. and foreman, U. S. Reclamation Service.
Present position: Foreman, U. S. Reclamation Service.

Alexander Watts McCoy, Ponca City, Okla.
Proposed by E. De Golyer, R. A. Conkling, Dorsey Hager.
Born 1889, Independnce, Mo. 1912, Univ. of Missouri, C. E., 1914, M. A. 1912–13, Engrg. Experiment Station, Univ. of Missouri. 1914–16, Instructor, Geol., Univ. of Oklahoma.
Present position: Geol., Morland Oil Company.

Richard Southgate McIntyre, Namtu, North Shan States, Burma.
Proposed by Andrew C. Lawson, Walter S. Weeks, T. A. Rickard.
Born 1895, Pittsburgh, Pa. 1910–12, Preparatory work, College of the City of New York, New York, N. Y. 1912–16, Univ. of California, B. S. 1915–16, Asst. Statistician, U. S. Geological Survey, San Francisco.
Present position: Asst. Min. Engr., Bowdwin Mine, Burma. Corpn., Ltd., Namtu, North Shan States, Burma.

Charles Mayott, Aurora, Nev.
Proposed by W. M. Dake, Jr., R. A. Hardy, F. A. Downes.
Born 1891, Salt Lake City, Utah. Grade Schools, etc., Salt Lake City. 1908–09, Mucker, Red Top Mine, Goldfield, Nev. 1909–11, Machine man, Timber man, Red Top Mine. 1911–13, Engrg. Dept., Goldfield Cons. Mines Co., Goldfield, Nev. 1913–14, Surface Boss, Aurora Cons. Mines Co., Aurora, Nev. 1914–15, Shift Boss. Aurora Cons. Mines Co., Aurora, Nev.
Present position: 1915 to date; Mine Supt., Aurora Cons. Mines Co.

Whitney Player Mee, Santa Rita, N. Mex.
Proposed by J. M. Sully, Horace Moses, Frank R. Wicks.
Born 1891, Salt Lake City, Utah. 1900–09, Public Schools, El Paso, Tex. 1909–13 Univ. of Cal., Berkeley, Cal., B. S. 1911, summer, Surveying, Mesilla Valley, U. S. R. S. 1912, summer, Mining, Copper Queen Mines, Bisbee, Ariz. 1913–14, Efficiency Engr., Oakland Lamp Works, of General Electric Co. 1914, Chem. and Asst. Engr., Shannon Copper Co., Gleeson, Ariz. 1914, Solution man, Simonds and Latham Cyanide Plant, Melones, Cal. 1914–15, Assayer, A. S. & R. Co., smelter, Asarco, Dgo., Mex. 1915, Car checker and jigger boss, Ray Copper Co., Ray, Ariz. 1915, Assayer, Asarco, Dgo., Mex.
Present position: 1915 to date; Chem., Chino Copper Co.

William Bruce Montgomery, Jr., Cleveland, O.
Proposed by J. A. Ede, Thomas T. Read, William H. Shearman.
Born 1880, Lynchburg, Va. 1900, Virginia Military Institute, Lexington, Va., B. S. 1909, School of Mines, Columbia Univ., E. M. 1901–06, Engr., Arlington C. & C. Co., Arlington, W. Va., and Gilliam C. & C. Co., Gilliam, W. Va. Shawnee C. & C. Co. and Glen Alum Coal Co., Glen Alum, W. Va. 1909–15, Engr., examination and report,work, The Grasselli Chemical Co., of Tenn., New Market, Tenn. Mgr. last year of operation. Since 1915, The Grasselli Chemical Co., Cleveland, O. Principally examination and report work, on zinc propositions.
Present position: Min. Engr., The Grasselli Chemical Co.

Edgar L. Newhouse, Jr., Salt Lake City, Utah.
Proposed by P. A. Mosman, C. W. Whitley, C. W. Stimpson.
Born 1890, Moncton, N. B., Canada. 1906–11, Columbia Univ., Min. & Met. 1912–14, Chem., and Assayer, East Helena Plant, American Smelt. & Ref. Co., Montana. 1914–16, Asst. Met., Garfield Smelt. Co., Utah.
Present position: Mgr., Garfield Chemical & Mfg. Corpn.

Chase S. Osborn, Sault Ste. Marie, Mich.
Proposed by O. C. Davidson, C. H. Baxter, George J. Eisele.
Born 1860, Huntington Co., Ind. LL.D. from several universities. Former Governor of Michigan, etc. 1883–1916, Chairman, Engineering Committee Board of Regents, Univ. of Mich. Field work in Lake Superior districts of Michigan, Wisconsin, Minnesota, Ontario, in Russia, Germany, Sweden, Lapland, Africa, Asia, Madagascar and South America, Cuba, China, etc., etc.
Present position: Explorer and Prospector.

Dale L. Pitt, Tacoma, Wash.
Proposed by Willard V. Morse, Brent N. Rickard, Roger E. Chase.
Born 1884, Salt Lake City, Utah. 1899–1903, Salt Lake High School. 1903–07, Utah School of Mines, B. S. 1907–09, Asst., J. W. Wade, M. E., Salt Lake City. 1909–12, Field Supt., B. F. Tibby Co., Salt Lake City. 1912–15, Chief Engr., Federal Lead Co., Flat River, Mo. 1915–16, Min. Eng., Salt Lake City.
Present position: Min. Eng., Tacoma Smelting Co.

Rastus S. Ransom, Jr., Golden, Colo.
Proposed by Charles F. Rand, Harry J. Wolf, W. G. Haldane.
Born 1890, New York, N. Y. 1907–09, Columbia Univ., School of Applied Science. 1910–13, Colorado School of Mines, E. M. 1913–14, Assayer, Bingham-New Haven Copper and Gold Min. Co., Bingham, Utah. 1914, Leasing and working underground, Central City and Idaho Springs Min. Districts, Colo.
Present position: Min. Engr., Golden, Colo.

Charles Henderson Rich, Conshohocken, Pa.
Proposed by John W. Logan, R. M. Bird, R. J. Wysor.
Born 1868, Yardley, Pa. Common school followed by technical training and special study. 1893–98, Analyst, Chemical Laboratory, Duquesne Works, Carnegie Steel Co. 1898–1901, Chem., Aliquippa Steel Co. 1901–02, Asst. Chem., Duquesne Works, Carnegie Steel Co. 1902–12, Chief Chem., Clairton Works, Carnegie Steel Co.
Present position: 1912 to date; Met., Alan Wood Iron & Steel Co.

Herman F. Schlundt, St. Francois, Mo.
Proposed by Waldo H. Comins, Philip H. Pipkin, James A. Caselton.
Born 1878, Rahway, N. J. 1899–13, Stevens Institute of Technology. 1903–04; 1905–06, Union Pacific R. R. 1906–07, Rock Island Lines. 1907–09, Taylor Gold Min. Co. 1909–11, Northwestern States Portland Cement Co. 1911–12, Cia. Metallurgica de Torreon. 1912–13, Amer. Smelt. & Ref. Co.
Present position: 1913 to date; St. Louis Smelt. & Ref. Co.

Ellwood Spencer Smith, Humboldt, Ariz.
Proposed by G. M. Colvocoresses, J. N. D. Gray, A. F. Bassett.
Born 1890, Brooklyn, N. Y. 1895–1902, Grade Schools, New York, N. Y. 1902–06, High School, New York, N. Y. 1906–10, U. S. Naval Academy, Annapolis. 1911, Post Graduate School, Univ. of Illinois. 1912, Clerk, International State Bank, Trinidad, Colo. 1914, Millman, Cons. Arizona Smelt. Co. 1915, Engrg. Staff, Cons. Arizona Smelt. Co.
Present position: Asst. to Genl. Mgr., Cons. Arizona Smelt. Co.

Frank J. Staral, Jr., Cleveland, O.
Proposed by S. H. Pitkin, Franklin Moeller, Joseph Struthers.
Born 1878, Akron, O. Akron and Cleveland Grammar Schools. 1899, Grad., Cleveland Central High School. 1902–05, Case School of Applied Science. 1904, Grad., Columbia School of Mines, E. M. 1904–06, Surveying Engr., The Boston and Montana Cons. Copper and Silver Min. Co., Butte, Mont., also experimental work at Great Falls Smelter. 1906–07, Draftsman and Surveying Engr., Cia Real del Monte y Pachuca, Pachuca, Hdg., Mex. 1907–13, Cashier, Min. Engr. and Supt., American Smelt. & Ref. Co., and allied interests in Mexico. 1913–14, Examination work and Mgr., The Assets Gold Min. Co., Jefferson City, Mont. 1914–16, Gaylord W. Feogo Co., Cleveland, O.
Present position: Supt., The Investors Co.

William Carl Strohbach, Hayden, Ariz.
Proposed by David D. Moffat, William T. MacDonald, David M. Kay.
Born 1891, San Francisco, Cal. 1896–1904, Public School, Oakland, Cal. 1904–08, Polytechnic High School, Oakland, Cal. 1909–14, Univ. of Cal., Berkeley, Cal., B. S. 1910–14, Summers, underground and cyanide plant, North Star Mine and Empire Mine, Grass Valley, Cal. 1915, Attendant at Bureau of Mines Exhibit, P. P. I. E., San Francisco, Cal.
Present position: 1915 to date; Chemist, Ray Cons. Copper Co.

Clarke Sullivan, Oakland, Cal.
Proposed by H. M. Alley, Robert G. Davies, E. E. Carpenter.
Born 1882, Atlanta, Ga. 1902, Oakland High School. 1907, Univ. of Cal., B. S. 1907-11 Assayer and Mill Supt., Progreso Min. Co. 1911–14, Mill Supt. and

Genl. Supt., Zopitoto Min. Co. 1914–15, Const. Supt., Mill work, Arrondo Min. Co. 1915, Mill work, Churchill Mill. Co.

Present position: Mill Supt., Eden Min. Co., Nicaragua, C. A.

E. L. Talbot, Salt Lake City, Utah.

Proposed by Alfred Frank, George H. Dern, Ernest Gayford.

Born 1862, Central City, Colo. Public grade school also High School. Daly West Min. Co., Park City, Utah. Ontario Silver Min. Co., Park City, Utah.

Present position: Genl. Supt., Daly West Min. Co., also Ontario Silver Min. Co.

Floyd Alpine Warren, Sr., Collinsville, Ill.

Proposed by Jesse O. Betterton, A. T. March, W. E. Newnam.

Born 1886, Hiawatha, Kans. 1904, Academy, Hiawatha, Kans. 1910, Highland Park College, Des Moines, Ia. 1910–15, Shift Boss, Foreman, Supt., Pattinson Plant Dept., Experimental work and Assaying, American Smelt. & Ref. Co., Omaha, Nebr.

Present position: Supt., Blast Furnace, Roasting Depts. and Newnam Hearths, St. Louis Smelt. & Ref. Co.

Sidney E. Werthan, Bisbee, Ariz.

Proposed by Gerald Sherman, Arthur Notman, H. C. Henrie.

Born 1891, Denver, Colo. 1910, Public Schools and High School, Denver, Colo. 1914, Univ. of Denver, B. A.; 1915, M. A. 1913–14, Instr., Dept. of Chem., Univ. of Denver. 1915, Research work on Flour, Great Western Mill. & Granary Co. 1915, Mine Laboratory, Copper Queen Cons. Min. Co.

Present position: Chemist, Copper Queen Cons. Min. Co.

George Hazzard West, Globe, Ariz.

Proposed by William B. Cramer, L. O. Howard, Walter Harris.

Born 1882, Philadelphia, Pa. 1896–1900, Central High School, Philadelphia, Pa., B. A. 1900–04, Univ. of Penn., Philadelphia, Pa., B. S. 1904–06, Assayer, Rincon Mines Co., Congress Junction, Ariz. 1906–07, Chem., Cons. Arizona Smelt. Co., Humboldt, Ariz. 1907–11, Chem. and Engr., Orogrande Smelt. Co., Orogrande, N. M. 1911 to date; Chem. and Smelter Foreman, Old Dominion Copper Min. & Smelt. Co., Globe Ariz.

Present position: Smelter Foreman, Old Dominion Copper Min. & Smelt. Co.

Henry Palmer Westcott, Pittsburgh, Pa.

Proposed by R. B. Woodworth, Thomas T. Read, William H. Shearman.

Born 1878, Seneca Falls, N. Y. High School at Seneca Falls and Rochester, N. Y. Five years, Alden Bateria National Gas Co., Alden, N. Y. 1908 to date; Metric Metals Works, Erie, Pa. Author, Hand Book of Natural Gas; Measurement of Gases Where Density Changes. Now in press, Hand Book of Casinghead Gas.

Present position: Engr., National Gas Construction.

Garratt S. Wilkin, Moscow, Utah.

Proposed by Edward R. Zalinski, Earl R. Pembrooke, C. W. Stimpson, W. A. Wilson, Ernest Gayford.

Born 1882, New Philadelphia, Pa. 1895–99, New Philadelphia High School. 1900–02, Western Reserve Univ. 1904, Miner, Katherine Mine, Stockton, Utah. 1905, Foot nipper, Bingham and New Haven Min. Co., Bingham, Utah. 1905–07, Sampler and Foreman, Boston Cons., Bingham, Utah. 1907–09, Supt., Utah Copper Co., Bingham, Utah. 1909–11, Supt., Wild Bill Min. Co., Milford, Utah.

Present position: 1911 to date; Supt. and Asst. Mgr., Moscow Min. & Mill. Co.

Charles F. Willis, Tucson, Ariz.

Proposed by Arthur Notman, Roger T. Pelton, Gerald F. G. Sherman.

Born 1885, Boston, Mass. 1902–06, Mass. Inst. of Tech., B. S. 1906–09, Assayer to Supt., Enterprise Min. Co., Cooney, N. M. 1909–10, Mgr., Pelican Min. & Mill. Co., Lake City, Colo. 1910–11, Mgr., Granite State Mica Co., Alstead, N. H. 1911, Prof. of Min. Engrg. and Director, Arizona State Bureau of Mines, Univ. of Arizona, Tucson, Ariz.

Present position: Director, Arizona State Bureau of Mines.

Harold Edwin Willson, Urbana, Ill.

Proposed by H. H. Stoek, C. M. Young, C. E. Krebs.

Born 1881, Lykens, Pa. 1900, Grad., Baltimore City College. 1916, Univ. of Ill., B. S. 1900–01, U. S. Fidelity and Guaranty Co., Baltimore, Md. 1901–13, Chainman, rodman, levellor, transitman, office man and chief engineer, McKell Coal

& Coke Co., Glen Jean, W. Va. 1904–13, Engr., McKell Heirs. 1905–13, Engr., Nichol Colliery Co. 1904–13, Engr., Laura Min. Co.
Present position: Student.

Frank A. Woodward, Cooper Hill, Ariz.
Proposed by B. Britton Gottsberger, P. G. Beckett, C. E. Mills.
Born 1852, Worcester, Mass. General education only. 1906 to date, Iron Cap Copper Co.
Present position: Supt., Iron Cap Copper Co.

Herbert Watson Woodward, Copper Hill, Ariz.
Proposed by W. B. Cramer, C. Mendelsohn, Walter Harris.
Born 1883, So. Framingham, Mass. 1900, Superior, Wis., Grammar and High Schools. 1900–02, High school, Somerville, Mass. 1903–06, Tufts College. 1906–08, Surveying mining claims. 1908, Assayer, Mine Surveyor, Copper Basin Min. Co. (defunct), Nev. 1908, Cyanide Mill, North Star Mine, Grass Valley, Cal. 1909–16, Engr., Iron Cap Copper Co., Copper Hill, Ariz.
Present position: Asst. Supt., Iron Cap Copper Co.

Yeh-Tze Yen, Hanyang, China.
Proposed by Chin Tao Huang, Cho Yang, C. F. Wang.
Born 1886, Kiangsu, China. 1901–03, St. John's College, Shanghai. 1904–07, Kyoto High School, Japan. 1907–10, Min. course, Kyoto Univ., Japan. 1911–12, Engr., Pinchuho Iron & Coal Co.
Present position: 1912 to date; Engr., Blast Furnace Dept., Hanyang Iron & Steel Works.

Stuart M. Young, Bingham, Utah.
Proposed by A. M. Day, J. A. Rule, A. B. Young.
Born 1890, Salt Lake City, Utah. 1896–03, Salt Lake Public schools. 1903–07, Salt Lake High School. 1907–11, Univ. of Utah, B. S. 1911–12, Chem., Grasselli Chemical Co., Park City, Utah.
Present position: 1912 to date; Assayer, Min. Dept., United States Smelt. Co.

Associate Members

William Wright Armstrong, Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Duncan MacVichie.
Born 1865, Darlington, Wis. 1885, Baker Univ., Baldwin, Kans., B. S. 1887, Univ. of Wisconsin, Madison, Wis., LL. B.
Present position: Pres., National Copper Bank.

Henry Ayman Bomberger, Philadelphia. Pa.
Proposed by Arthur J. Hoskin, F. W. Smith, J. W. Finch.
Born 1866, Philadelphia, Pa. 1884, Ursinus College, Pa., A. B.; 1886, A. M. 1900, Vice-Prest., Temple Univ., Philadelphia. 1905, Prest. & Genl. Mgr., Southwestern Dev. Co. 1912, Genl. Mgr., Minnesota-Connor Min. & Mill, Co. Have had no special technical training in the schools as a mining engineer. Have been a student all my life and for nearly fifteen years have made a special study of mining in all its branches, about one-third of which time has been spent in practical field work associated with some of the leading mining engineers and mine managers of the country, none of whom will know that I have ever made any claims at all as an engineer for I have never made any such claims. I personally organized the three companies above mentioned, and my services for mine reports, etc., are constantly sought without solicitation on my part.
Present position: Prest. & Genl. Mgr., Mohave Minerals Co.

William Paul Gage, Fort Worth, Tex.
Proposed by Samuel S. Wyer, Harry C. Reeser, David T. Day.
Born 1875, Denver, Colo. 1894, Grad., Riverside (Cal.) High School, scientific course. 1895–96, Finished Sophomore year at Marietta (Ohio) College; Latin, scientific course. 1895, summer, and from spring, 1896 to spring, 1903, Eureka Pipe Line Co. and Buckeye Pipe Line Co. in W. Va. and Ohio. Worked in all the pipe line departments on construction of lines and stations, office and field work of all kinds. 1903–09, Security Oil Co., Beaumont, Tex.; Inventory Clerk; Chief Dispatcher; Asst. to Genl. Supt., Pipe Line Dept. 1909–16, Vice-Prest. and Genl. Supt., Lone Star Gas Co., Fort Worth, Tex., active executive officer. Practical management of construction and operation almost 500 miles pipe line, supplying about

30 towns in North Texas in addition to development of the gas field, installation of compressor station, etc.
Present position: First Vice-Prest. and Genl. Supt., Lone Star Gas Co.

Charles A. Gillette, Salt Lake City, Utah.
Proposed by C. W. Stimpson, Ernest Gayford, Walker J. Boudwin.
Born 1869, Sioux City, Iowa. 1891, B. S., Washington & Jefferson. 1893, Law, University of Iowa.
Present position: Attorney at Law.

William Joseph Quigly, El Paso, Texas.
Proposed by William D. Gordon, William S. Noyes, W. N. Small.
Born 1872, Carbondale, Pa. 1896–1900, Ibex Mining Co., Leadville, Colo. 1900–02, Penn M. & L. Co., Leadville, Colo. 1902–06, Chihuahua Min. Co. 1906–12, Genl. Agent, Chihuahua Min. Co. and El Potosi Min. Co., Chihuahua, Mex.
Present position: 1912 to date; Genl. Mgr., El Potosi Min. Co., and Chihuahua Min. Co. and Director of both companies.

Henry Denison Randall, Salt Lake City, Utah.
Proposed by C. W. Whitley, Ernest Gayford, C. W. Stimpson.
Born 1881, Ledyard, Conn. 1902, Amherst College, B. S. 1902–03, Lawyer, Mann Electric Co. 1903–08, Westinghouse Electric Mfg. Co. 1908–12, Allis-Chalmers Mfg. Co.
Present position: 1912 to date, Mgr., General Electric Co.

James B. Walker, Jr., Salt Lake City, Utah.
Proposed by C. W. Stimpson, O. J. Salisbury, Ernest Gayford.
Born 1889, Helena, Mont. 1911, Cornell Univ., B. L., Mech. Engrg.
Present position: Mgr., Kelly Filter Press Co.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period June 10, 1916 to July 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 115, February to July, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of July 10, 1916.

Alderson, Victor C., Cons. Engr., Winnemucca Mountain Min. Co., Winnemucca, Nev.
Allen, Arthur P. .... 233 College Ave., Houghton, Mich.
Anderson, A. J. .... Care Moose Mountain, Ltd., Sellwood, Ont., Canada.
Anderson, Ray B. .... 11 Cliff St., New York, N. Y.
Babb, P. A., Care Blaisdell Coscotitlan Syndicate, Apartado No. 92, Pachuca, Hidalgo, Mexico.
Bard, D. C. .... 660 Stuart Bldg., Seattle, Wash.
Bassett, Thomas E. .... 59 Auburn Ave., Pontiac, Mich.
Batcheller, J. H., Care Virginia Lead & Zinc Corpn., Jone's Store P. O., Spottsylvania Co., Va.
Bennett, A. F. .... 2 Rector St., New York, N. Y.
Binford, C. M. .... J. B. B. Coal Co., Twin Branch, W. Va.
Bradt, Harlan H. .... 709 Alworth Bldg., Duluth, Minn.
Brandes, Juan Felix ...... 139 Whittell Bldg., 166 Geary St., San Francisco, Cal.
Brayton, Corey C., Min. Engr .... Hobart Bldg., San Francisco, Cal.
Brenneman, F. G. .... 105 N. George St., Pottsville, Pa.
Brennon, J. C. .... Room 868, 11 Broadway, New York, N. Y.
Brooke, Lionel, ... Care Morococha Mining Co., Morococha, Peru, S. Amer.
Brunton, Frederic K. .... Garfield Smelter, Garfield, Utah.
Bryan, J. K. ...... Care Electrolytic Zinc Co., Colgate P. O., Baltimore, Md.
Burch, H. Kenyon, Care The Sierra Madre Club, L. A. Investment Bldg., Los Angeles, Cal.
Callaway, L. A. .... 512 Pine St., Anaconda, Mont.
Campbell, W. C. .... 705 South Main St., Salt Lake City, Utah.
Carson, A. C. .... 800 Bush St., San Francisco, Cal.
Claghorn, D. M. .... Care Morning Hotel, Mullan, Idaho.
Cook, Paul R. .... Box 655, Guayaquil, Ecuador, S. Amer.

Cooner, John D.............................417 Wyoming Ave., Scranton, Pa.
Cornelissen, Johannes......................Santa Marta, Colombia, S. Amer.
Coxe, Edward H., Care United Coal Corpn., First National Bank Bldg., Pittsburgh, Pa.
Cullum, J. Barlow......................Elaridge, Leetsdale, Allegheny Co., Pa.
Dewey, F. P......................832 Varnum St., N. W., Washington, D. C.
Dowd, James J....................................Hotel Dee, Houghton, Mich.
Drummond, T. R.................Hongkong and Shanghai Bank, Manila, P. I.
Earle, Theodore..................................165 Broadway, New York, N. Y.
Ellis, Ernest W.......Care Northern Ore Co., Edwards, St. Lawrence Co., N. Y.
Enzian, Charles, Philadelphia and Reading Coal & Iron Co., Pottsville, Pa.
Estabrook, E. L. Geol..................Wisconsin Zinc Co., Platteville, Wis.
De Faria, Cicero Coelho, Engr. Fiscal das Estradas, Inspectoria Federal das Estradas, Rua Conde do Bomfim, No. 46, Rio de Janeiro, Brazil, S. Amer.
Fearing, F. C., Cons. Engr., Kelly, Cooke & Co., Drexel Bldg., Philadelphia, Pa.
Fearn, Percy LeR..............................52 William St., New York, N. Y.
Feldenheimer, Roy..Moberly, Mo.
Fischer, Siegfried, Jr.................Care Mrs. C. E. Parfet, Golden, Colo.
Fisher, Thomas E.......................112 East 17th St., New York, N. Y.
Fletcher, A. R.......................................Box 540, Palo Alto, Cal.
Foester, H. W...........Care Lucky Tiger Min. Co., Esqueda, Sonora, Mexico.
Fraser, William L., Supt...Oro Grande Mining Co., Callahan, Siskiyou Co., Cal.
Gold, Charles B., Chemist and Assayer, Bartlesville Zinc Co., Collinsville, Okla.
Gordon, Robert, Mine Supt., Abangarez Gold Fields, Abangarez, Costa Rica, C. A.
Gormly, S. J., Min. Engr............................Guayacan, Chile, S. Amer.
Grieve, Richard P.............Narcozari, Son., Mexico, via Douglas, Ariz.
Grunow, William R., S'g't. Co. "M," Arizona 1st Infantry, Camp Harry J. Jones, Douglas, Ariz.
Hall, C. K.,.................................1722 West 28th St., Cleveland, O.
Hall, William J...........Care Federal Min. & Smelt. Co., Wallace, Idaho,
Hannahan, M. L..........................28 George St., Charleston, S. C.
Harder, E. C.......School of Mines, Univ. of Minnesota, Minneapolis, Minn.
Harley, G. Townsend, Efficiency Engr., Burro Mountain Copper Co., Tyrone, N. M.
Haseltine, R. S......................................Hermosa Beach, Cal.
Havlin, T. N............................6417 Stewart Ave., Chicago, Ill.
Head, James L., Engr..................Doe Run Lead Co., Rivermines, Mo.
Heller, M. J............................120 Broadway, New York, N. Y.
Herzig, Charles S., Mine Valuer, Consulting Practice, 27 William St., New York, N. Y.
Hilby, George R..Ruth, Nev.
Hobart, E. Norris..........................P. O. Box 621, Nogales, Ariz.
Housholder, E. Ross.......Care Nevada-Arizona Mines Co., Hackberry, Ariz.
Igawa, Tokeo..................Kojiromura, Kugagun Yamaguchiken, Japan.
James, Alfred.................28 Victoria St., Westminster, London, England.
Jansen, P., Genl. Mgr., Mynbouw Maatschappy Siman, Post Ketaun, Benkoelen, Sumatra., D. E. I.
Jennings, Robert E., 2d........................50 Oriental St., Newark, N. J.
Jensen, Joseph..................................Box U, Santa Fe, N. M.
Johnson, Guy R., Jr......................Florence Iron Co., Florence, Wis.
Keep, Glenn A............................1334 West 102d St., Cleveland, O.
Kellogg, L. O., Supt., South American Development Co., Box 655, Guayaquil, Ecuador, S. Amer.
Kinney, S. P.........................414 Boston Bldg., Salt Lake City, Utah.
Kuang, Y. C.,...Care Sun Quong Hing Co., Wing Lock St., Hong Kong, China.
Ladoo, Raymond B......................The Low Moor Iron Co., Low Moor, Va.
Laird, George A...North Yakima, Wash.
Laudig, O. O., Mgr..............Standard Iron Co., Deseronto, Ont., Canada.
Lawton, N. O...Lawton, Mich.
Lednum, E. T., Mgr., E. I. DuPont de Nemours & Co., Suite 808 Frisco Bldg., Joplin, Mo.
L'Engle, E. Fleming, Mgr.............................Royal Zinc Co., Joplin, Mo.
Levis, Alfred C......................................Box 578, Golden, Colo.
Lewis, Arthur H., Min. Engr...................G. B. Markle Co., Jeddo, Pa.
Little, James E........The Spanish American Iron Co., So. Bethlehem, Pa.
Loerpabel, W. Harrison.............Care Carlota Mine, Cumanyaqua, Cuba.
Lunn, Robert, Jr...Jerome, Ariz.

MacGregor, Frank S., Development Dept., E. I. DuPont de Nemours Co.,
452 DuPont Bldg., Wilmington, Del.
Mackey, R. W., Safety Engr. & Mine Inspector, Homestake Min. Co., Lead, S. D.
Magee, James F. .... Hold publications.
Mann, William Seward .... 2215 Como Ave., St. Paul, Minn.
Marshall, Holman Thompson .... American Fork, Utah.
Matteson, W. G., Geol., Empire Gas & Fuel Co., Box 578, Bartlesville. Okla.
Maycumber, W. R., Supt., Crushing & Tailings Disposal, Chile Exploration Co.,
Chuquicamata, Chile, via Antofagasta.
Milton, M. C. .... Box 716, Tucson, Ariz.
Mitchell, LeRoy B. .... Organ, N. M.
Murphy, E. M. .... 1720 E. 12th Ave., Spokane, Wash.
Murray, James J. .... Orogrande, N. M.
Nahl, Arthur C. .... 1079 Monadnock Bldg., San Francisco, Cal.
Neustaedter, H. A. .... Mineral, Va.
Nighman, C. Edwin .... 1103 Lewisohn St., Butte, Mont.
Olyntho, dos Santos Pires Antonio, 8 rua Leite Leal, Rio de Janeiro,
Brazil, S. Amer.
Otterson, George W. .... 728 South Fountain Ave., Springfield, O.
Owen, T. M., Genl. Mill Supt., Federal Mining & Smelting Co., Wallace, Idaho.
Peterson, Frank .... Hold Publications, P. O. N.
Pickering, J. C., Min. Engr. .... 17 Battery Place, New York, N. Y.
Pickett, George B. .... Nucla, Colo.
Pitman, S. M. .... Isle of Springs, Maine.
Porter, Fred S., Care Canadian-Klondike Min. Co., Dawson, Yukon Territory, Can.
Probert, Frank H. .... Min. Dept., Univ. of Cal., Berkeley, Cal.
Reece, P. P. .... Melcher, Marion Co., Iowa.
Reynders, J. V. W. .... Room 2218, 120 Broadway, New York, N. Y.
Robbins, Hallet R. .... 616 Seneca St., Seattle, Wash.
Ross, H. W., Asst. Mgr., Care Backus & Johnston Co., Casapalca, Peru, S. Amer.
Ruebel, E. H., Jr. .... 413 W. Monroe St., Springfield, Ill.
Sakikawa, M. .... 260 Shirogane Sankocho, Shibaku, Tokyo, Japan.
Sauerschnig, Jose .... Isabel La Cutolica 39, Mexico City, Mex.
Schindler, D. F. .... Care Vermont Copper Co., So. Strafford, Vt.
Scovil, H. H. .... Delta Upsilon House, So. Bethlehem, Pa.
Sielaff, G. J. .... 2010 Telegraph Ave., Oakland, Cal.
Sizer, Frank L. .... First National Bank Bldg., San Francisco, Cal.
Smith, Elwyn L. .... Freeland, Idaho Springs, Colo.
Smith, Frank G. D. .... U. S. Metals Ref. Co., East Chicago, Ind.
Smith, Lyon .... Apt. 2, 6226 Kimbark Ave., Chicago, Ill.
Spangler, Howard .... 1337 York St., Denver, Colo.
Staver, W. H. .... 505 Krise Bldg., Lynchburg, Va.
Struthers, Joseph .... 111 Broadway, New York, N. Y.
Sutton, Harry A. .... Aumsville, Ore.
Tainter, F. S. .... 60 Wall St., New York, N. Y.
Talcott, M. Gardner .... 31 Hewlett St., Waterbury, Conn.
Thomson, Edward .... 535 Grayson St., San Antonio, Texas.
Titsworth, F. S. .... 404 Equitable Bldg., Denver, Colo.
Trauerman, Carl J. .... 832 Colorado St., Butte, Mont.
Treat, L. B. .... 31 Commercial Union Bldg., Montreal, Can.
Tsai, Hsiang .... Care Oliver & Co., Hankow, China.
Van Zwaluwenburg, A., Chemist .... Nipissing Mines, Cobalt, Ont., Canada.
Vogel, Herman H. .... Mascot, Tenn.
Walters, M. B. .... P. O. Box 647, Rossland, B. C., Canada.
Weiberg, E. B. .... Care Pieher Lead Co., Henryetta, Okla.
Wentworth, H. A. .... 55 Congress St., Boston, Mass.
Williams, P. T. .... 2214 Union St., Berkeley, Cal.
Williams, Ralph B. .... Room 2202, 120 Broadway, New York, N. Y.
Yatsevitch, M. G., Care Prof. N. T. Stepanoff, Chemical Lab., Min. Institute,
Vasilieff Ostrov. 21 Line, Petrograd, Russia.

MEMBERS' ADDRESSES WANTED

Name.	Last Address of Record from which Mail has been Returned.
Barnes, Blakeslee	Arrow Engineering Co., Palmyra, Mo.
Bell, D. A. S.	136 McLaren St., Ottawa, Canada.

BLOW, J. J. .... 172 Rodney St., Brooklyn, N. Y.
BOOTH, E. L. .... 349 W. 145th St., New York, N. Y.
BOYS, H. R. .... Aurora Cons. Min. Co., Aurora, Nev.
CAIRNS, JOHN M. .... Henry S. King & Co., 65 Cornhill, London, E. C., England.
COLE, ROBERT J. .... McKay Apartments, 7th & Pike Sts., Seattle, Wash.
CRARY, CHARLES N. .... Kimberly, Nev.
DRAPER, CARL H. .... Apartado 77, Guadalajara, Jal., Mexico.
FOSTER, GEORGE C. .... 18 Cadogan Blk., Calgary, Alta., Canada.
GOEDICKE, CARL .... Box 535, San Antonio, Tex.
GORDON-FIREBRACE, W. E. .... 812 Salisbury House, London, E. C., England.
GRANT, ULYSSES S., IV .... Westmorly Court, Cambridge, Mass.
GUNTHER, C. GODFREY .... Stratford, Conn.
HANLON, JOHN EDWARD .... Timmins, Ont., Canada.
HOBART, EDMUND NORRIS .... Clifton, Ariz.
JONES, THOMAS J., Mine Mgr., Kyshtim Min. Wks., Perm Govt.,
via Petrograd, Russia.
KELLOGG, R. M. .... Wellington, Lyon Co., Nev.
KISHMAN, MAURICE W. .... 101 Masonic Ave., Cripple Creek, Colo.
LENOIR, FRANK H. .... Douglas, Alaska.
MCGEE, JOHN .... United Greenwater Co., Dale via Amboy, Cal.
MAINWARING, H. M. C., Mine Mgr., Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MILLER, FRANK BARTON .... Blair, Esmeralda Co., Nev.
MOHRMAN, E. M. .... 1293 W. 111th St., Cleveland, O.
MOORE, REDICK R. .... Zortman, Mont.
O'BRIEN, P. C. K. .... Hotel Essex, 684 Larkin St., San Francisco, Cal.
PARRISH, S. F. .... Battle Mountain, Nev.
PATCHELL, F. J. .... 4516 No. Lincoln St., Chicago, Ill.
PATERSON, A. W. .... 1814 11th Ave., Spokane, Wash.
PERRY, ROBERT S. .... Kingscourt Apts., 36th & Chestnut Sts., Phila., Pa.
RALPH, E. W. .... Boston Ely Min. Co., Kimberly, Nev.
REDFEARN, A. M. .... Hold publications.
REVELL, G. E. .... Box 132, Nelson, B. C., Canada.
REYNOLDS, L. .... Apartado 25, Guauajuato, Mex.
RHODES, W. B. .... Golden, Colo.
RODRIQUEZ, J. C. .... Apartado, 87 Saltillo, Coah., Mexico.
ROGERS, B. C., Care Standard Mine, Detroit Copper Min. Co., Box 427,
Metcalf, Ariz.
SALES, A. J. .... Giroux Cons. Mines Co., Kimberly, Nev.
SANDIFER, H. C. .... Box 15, Bis., Mexico City, Mexico.
SMITH, A. H. .... Gateway, B. C., Canada.
SPARKS, J. T. .... 326 Erie St., El Paso, Tex.
STODDART, A. W. .... 638 Salisbury House, London, E. C., England.
SULLIVAN, W. P. .... The Henry Walke Co., Norfolk, Va.
TAYLOR, A. W. .... Korean Exploration Co., Chiksan Mines, Chiksan, Korea.
THOMAS, EDMUND .... Weaver Mine, Gibson, N. Mex.
TOD, GRANT H. .... Care Coast Mfg. & Supply Co., Livermore, Cal.
TONG, S. K. .... 413 W. 115th St., New York, N. Y.
VAN NESS, W. W. .... 622 Salisbury House, London, E. C., England.
WAINEWRIGHT, W. B. .... 61 Grace Church St., London, E. C., England.
WARD, HARRY J. .... 948 Market St., San Francisco, Cal.
WENTWORTH, I. H. .... 245 Beldon Ave., Harlandale Addition, San Antonio, Tex.
WRAIGHT, E. C. .... 63 Wavertree Road, Streatham Hill, London, S. W., England.
YEANDLE, W. H., JR., Care Guilermo Brockmann,
2a Capuchines, No. 55, Mexico City, Mex.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period June 10, 1916 to July 10, 1916.

Date of Election.	Name.	Date of Decease.
1888	*Gatzmer, William C.	May 2, 1916.

* Member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.
DAVID H. BROWNE, *Chairman.* PERCY E. BARBOUR, *Vice-Chairman.*
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer.*

Boston

Meets first Monday of each winter month.
W. E. C. EUSTIS, *Chairman.* R. L. AGASSIZ, *Vice-Chairman.*
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.
STANLY A. EASTON, *Chairman.* FREDERIC KEFFER, *Vice-Chairman.*
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
D. C. LIVINGSTON, FRANK A. ROSS.

Puget Sound

Meets second Saturday of each month.
GLENVILLE A. COLLINS, *Chairman.* H. L. MANLEY, *Vice-Chairman.*
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

Southern California

C. COLCOCK JONES, *Chairman.* ALVIN B. CARPENTER, *Vice-Chairman.*
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Bldg., Los Angeles, Cal.
A. B. W. HODGES, R. A. PEREZ,
E. A. MONTGOMERY, WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman.* F. H. BOSTWICK, *Vice-Chairman.*
P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
G. A. KENNEDY. M. S. MACCARTHY.

Montana

J. L. BRUCE, *Chairman.* W. C. SIDERFIN, *Vice-Chairman.*
M. H. GIDEL, *Secretary-Treasurer*, 1102 W. Galena St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.
T. A. RICKARD, *Chairman.* W. H. SHOCKLEY, *Vice-Chairman.*
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
E. A. HERSAM, H. W. YOUNG.

Pennsylvania Anthracite

R. V. NORRIS, *Chairman.*
CHARLES F. HUBER, *Vice-Chairman.* EDWIN LUDLOW, *Vice-Chairman.*
W. J. RICHARDS, *Vice-Chairman.* ARTHUR H. STORRS, *Vice-Chairman.*
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP,
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI. *Chairman.* HERMAN GARLICH, *Vice-Chairman.*
F. W. DEWOLF, *Vice-Chairman.* M. M. VALERIUS, *Vice-Chairman.*
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH,

Chicago

CHARLES H. MACDOWELL, *Chairman.* LUTHER V. RICE, *Vice-Chairman.*
HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

Utah

C. W. WHITLEY, *Chairman.* WALTER FITCH, *Vice-Chairman.*
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI, WILLIAM WRAITH.

Arizona

GERALD SHERMAN, *Chairman.*
NORMAN CARMICHAEL, *1st Vice-Chair.* B. BRITTON GOTTSBERGER, *2nd Vice-Chair.*
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
W. L. CLARK, J. C. GREENWAY.
W. G. McBRIDE. FOREST RUTHERFORD.

Nevada

J. W. HUTCHINSON, *Chairman.* FRANCIS CHURCH LINCOLN, *Vice-Chairman.*
HENRY M. RIVES, *Secretary-Treasurer*. Reno, Nevada.
W. H. BLACKBURN, E. A. JULIAN,
EMMET D. BOYLE, JOHN G. KIRCHEN,
FREDERICK BRADSHAW, C. B. LAKENAN.
TASKER L. ODDIE.

STANDING COMMITTEES

Executive

L. D. RICKETTS, *Chairman.*

GEORGE D. BARRON,
SIDNEY J. JENNINGS,
W. L. SAUNDERS,
BENJAMIN B. THAYER

Membership

KARL EILERS, *Chairman.*

ARTHUR S. DWIGHT,
LEWIS W. FRANCIS,
LOUIS D. HUNTOON,
ARTHUR L. WALKER.

Finance

GEORGE D. BARRON, *Chairman.*

ALBERT R. LEDOUX,
CHARLES F. RAND.

Library

E. GYBBON SPILSBURY, *Chairman.*[3]

KARL EILERS,[1]
ALEX C. HUMPHREYS,[2]
E. F. ROEBER,[4]
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Features of the New Copper Smelting Plants in Arizona

BY A. G. MCGREGOR,* B. S., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

DURING the past 5 years, five new copper-smelting plants have been built and put into operation in the State of Arizona. The monthly copper output from these plants averages from 5,000,000 to 18,000,000 lb. Previously, there never was as much activity in copper-smelting plant construction in the same length of time in all the rest of the world.

Naturally, in this amount of work, some new problems were met and new features in plant design and equipment developed. Some of these features are described in this paper.

CONCENTRATE HAULING, UNLOADING AND SAMPLING SYSTEM

(a) *Concentrate Car*

The International Smelting Co. has provided at its plant at Miami specially designed cars for hauling the concentrates from the mills of the Inspiration Copper Co. and the Miami Copper Co. The car has a bottom that is readily made tight to hold flotation concentrates and is quickly and cheaply unloaded. Figs. 1, 2 and 3 show views and details of the car.

The main feature of this car is the slot in the bottom throughout its length, which is closed by means of short planks that overlap like shiplap as shown by Fig. 1. At one end of the car an iron gate operated by a screw and hand wheel is provided, which clamps or presses the planks together, making the bottom tight. A removable tapered plug rests vertically over this gate when the loading of the concentrates at the mill commences.

A concentrate unloading pocket (Fig. 4) 180 ft. long, is provided at the smelting plant, which has a conveyor belt running throughout its length underneath the track. To unload a car, the vertical plug at the end of the car is lifted by means of a chain hoist which is supported by a trolley overhead. The gate at the end of the car is opened approximately 18 in. The removal of the plug leaves a hole down through the concentrates. The concentrates are poked through the hole from the top of the car until it is convenient for a man to stand on the bottom and draw one of the slats in the bottom toward the gate end of the car. The concentrates are then raked down with hoes on their natural angle of repose,

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* Mechanical and Metallurgical Engineer.

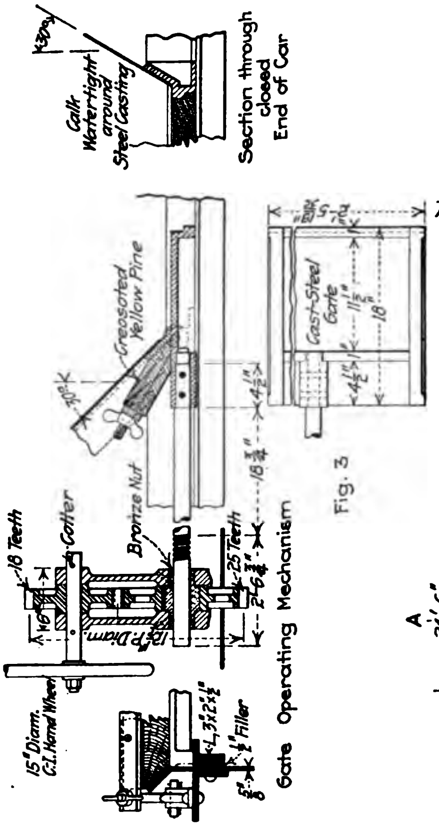
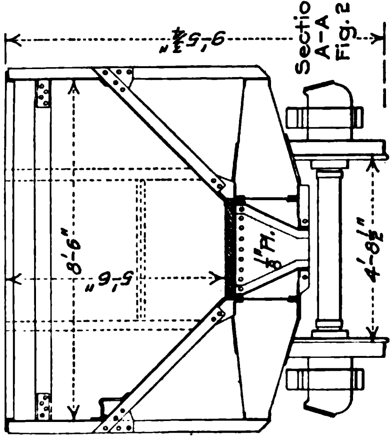


Fig. 3



Section A-A Fig. 2

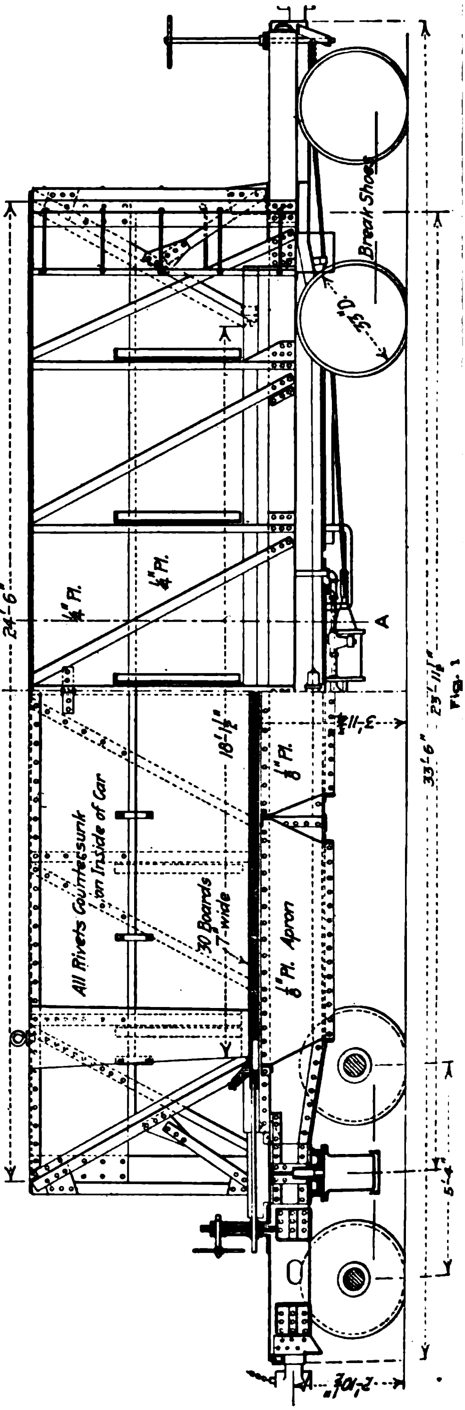


Fig. 1

which is nearly vertical, until the next slat is partly uncovered, when it is pulled toward the gate end of the car. This operation progresses until the opposite end of the car is reached. The slats and the grooves supporting them are then carefully swept off and cleaned with a broom; the gate is screwed back, pressing the slats together, and the plug is lowered into place. The car is then ready to be taken back to the concentrator for another load.

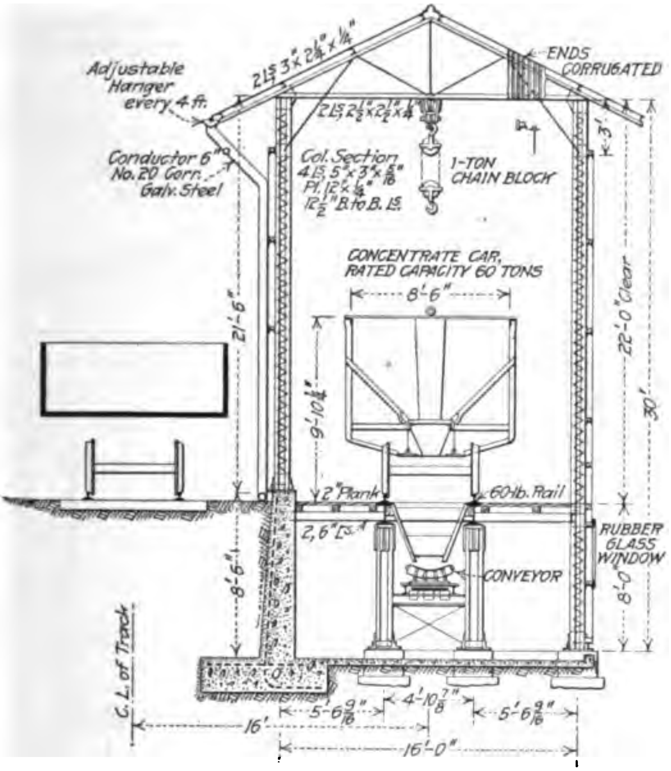


FIG. 4.—SECTION THROUGH CONCENTRATE UNLOADING POCKET.

(*b*) *Concentrate Sampler*

Fig. 5 shows diagrammatically the scheme used for sampling the concentrates. It will be noted that the conveyor from the unloading pocket discharges onto a shuttle conveyor (a tripper on the former conveyor could have been substituted for the shuttle conveyor but it would not have handled the sticky concentrates as well). Buckets attached at their ends to chain belts are arranged to cut through the stream of concentrates as they are discharged onto the shuttle conveyor. The chains are driven and supported by sprocket wheels. The arrangement is such that at the highest point in their travel the buckets are in an inverted position. At this point, they pass under a revolving shaft which has a number of knockers, made of 6-in. belting, attached to it. These knockers slap the bottom of the buckets, jarring loose any concentrates tending to stick to them. This arrangement works satisfactorily on ordinary table and vanner concentrates, but is not satisfactory with flotation concentrates on account of the varying load on the conveyor belt, the flotation concentrates being unloaded from the car in large chunks.

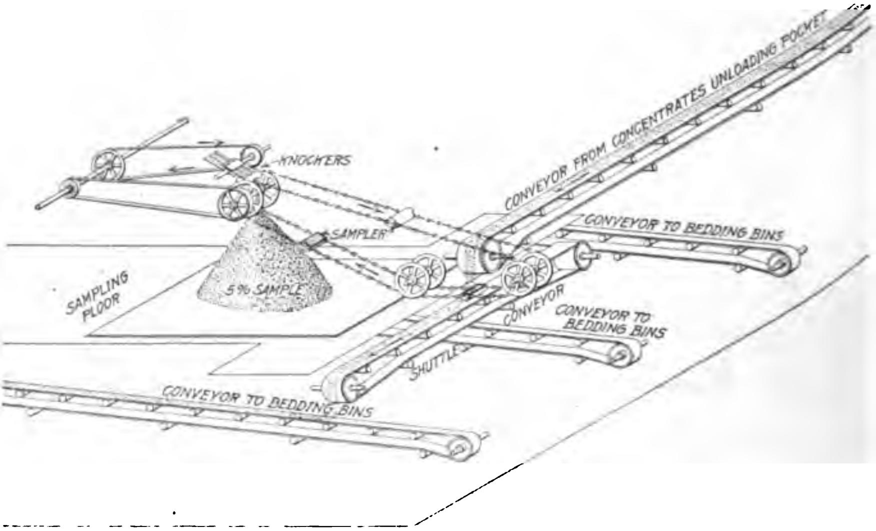


FIG. 5.—ARRANGEMENT FOR SAMPLING CONCENTRATES.

(c) *Bedding Bins*

Fig. 6 shows a cross-section of the bedding bins used in connection with the concentrate car and sampler just described. The bins are 150 ft. long and have a total capacity of 9,000 tons of charge. The concentrates are delivered by belts from the car-unloading pocket to any one of the three conveyors passing over the bins. Each belt has a motor-operated tripper which travels back and forth throughout the length of the bin. Each tripper is arranged with two cross-belts, which discharge material on both sides of the tripper, instead of the ordinary chutes that would clog with the sticky concentrates. These cross-belts also permit of a wider top on the ore bed than could be obtained with ordinary chutes. The cross-belts of the tripper as well as the long belts running over the bins and all the other belts handling sticky concentrates are speeded to 400 ft. per minute, which aids in discharging the sticky material from the belt when it passes over the pulley at its head end or the pulley of the tripper. An oscillating deflector is so arranged that the material passing over the tripper is discharged, first on one of the short cross-belts and then on the other. This is done to provide an even distribution of the material over the top of the beds, making the height of the bed on both sides of the tripper the same. The tripper is reversed at each end of its travel by the reversal of its propelling motor through a magnet-switch control. The bins have "V" bottoms with a slot 2 ft. 4 in. wide running throughout their length. This slot is covered with short planks which are readily removed, and as the reclaiming progresses, they are merely moved back in the slot.

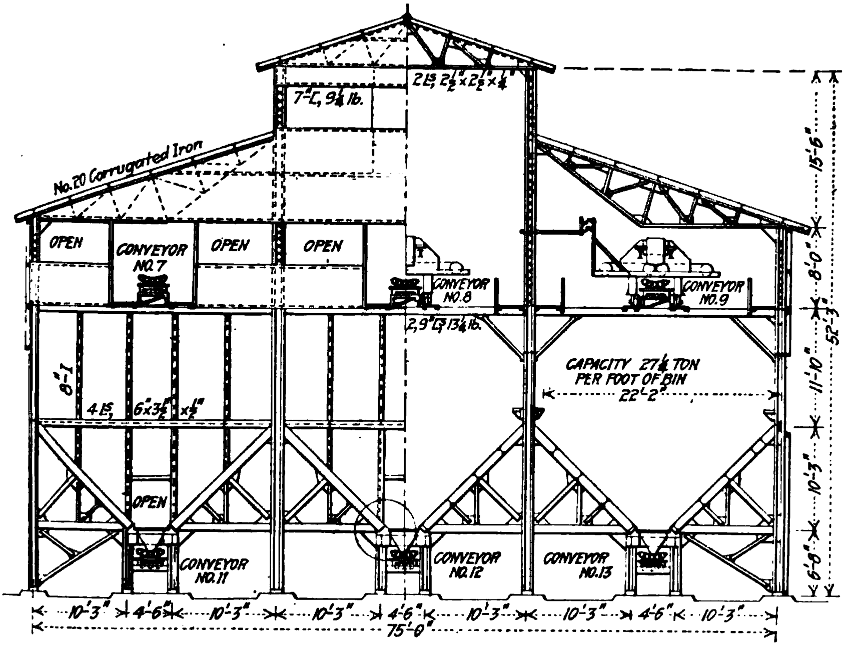


Fig. 6.—Cross-Section of Bedding Bins.

The concentrates are raked down with hoes and allowed to discharge, through the opening in the bottom thus provided, onto the conveyor belts underneath. The concentrates are bedded in these bins with a proper amount of limestone and pyrite and the available secondaries, each bin comprising a bed. Storage bins for storing finely crushed lime rock, secondaries, pyrite, etc., are provided and arranged so that the fine material from these bins is delivered by a conveyor and discharged just ahead of the concentrates onto the belts leading up over the bedding bins. The material next to the bedding belts, therefore, consists largely of finely crushed lime rock, which is a further aid in the discharge of the sticky concentrates from these belts.

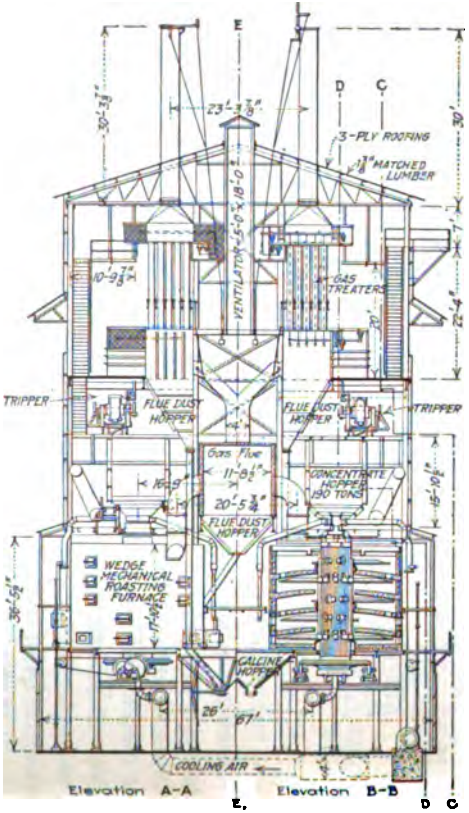


Fig. 7.—End Elevation of Roaster or Dryer Plant, International Smelting Co.

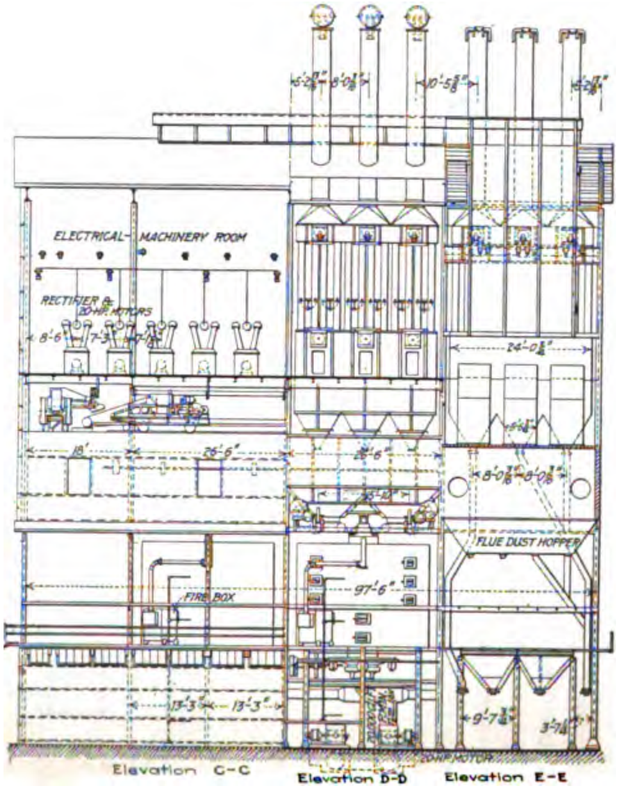


Fig. 8.—Side Elevation of Roaster or Dryer Plant, International Smelting Co.

Roaster or Dryer Plant

Figs. 7 and 8 show end and side elevations respectively of the International Smelting Co.'s roaster or dryer plant. The main features of this plant consist in the method used for conserving the heat and in the placing of the Cottrell fume treaters directly above the furnaces. The roaster furnaces in this plant are used for drying the concentrates and are not used for roasting, as the charge contains no excess sulphur; in fact it has been necessary to have some pyrite shipped in from Bisbee in order to raise the sulphur content of the charge sufficiently to prevent the production of mattes too high in copper for good clean slags.

(a) *Conservation of Heat in Furnace*

The furnaces are air-cooled. The cooling air, after passing through the furnace arms, is conducted up the central shaft to the top, thence down to the oil fireboxes connecting with the two lower hearths of the furnace. This gives preheated air for the combustion of the oil and conserves the heat absorbed by the cooling air. To further conserve the heat of the furnace, the linings are 12 in. thick. The outer 4 in. of lining, between hearths, is made of "Nonpareil" insulating brick. The manufacturers of these brick claim that 4 in. of their brick is equivalent in heat insulating value to 40 in. of ordinary brick. The tops of the calcine hoppers of these furnaces are insulated from the lower floor of the furnace building, just above them, by an air space. The calcine hoppers have non-conducting linings consisting of 2½ in. of Nonpareil insulating brick and a 1-in. layer of reinforced concrete. The reinforced concrete is to take the wear and erosion caused by the heated material inside the hopper, as the Nonpareil brick will not stand much wear or rough usage.

(b) *Cottrell Dust-collecting System for Furnaces*

The Cottrell equipment for treating the gases from this plant consists of 12 units of 20 tubes each. The tubes are of lap-welded steel 13-in. in outside diameter, and 15 ft. in length. They are flanged outward, or "Vanstoned" at each end, which reduces the brush discharge caused by sharp corners and makes a good means of connection to the upper and lower diaphragms. Suitable lugs are welded to the pipes against which the hammers strike when vibrating the tubes to shake off the collected dust. The electrical equipment for this installation is so arranged that the voltage used may be varied from 50,000 to 100,000 volts. The treater tubes were planned for a velocity of 5 ft. per second for the gases passing up through them. The recovery of dust by this treater is practically perfect.

The main feature of this installation is in its arrangement which permits the gases to pass directly up from the furnaces through the header flue, the tubes and the stacks, the gases rising continuously from the furnace hearths to the outlet of the stacks.

Blast Furnaces

Fig. 9 shows a general end elevation of the blast furnaces at the Calumet & Arizona Mining Co.'s plant at Douglas. These furnaces are 40 ft. long by 4 ft. wide at the tuyères. The ores and materials smelted in these furnaces are bedded so that the charge coming to the charge bins over the furnaces is thoroughly mixed. A coke bin and an ore bin are pro-

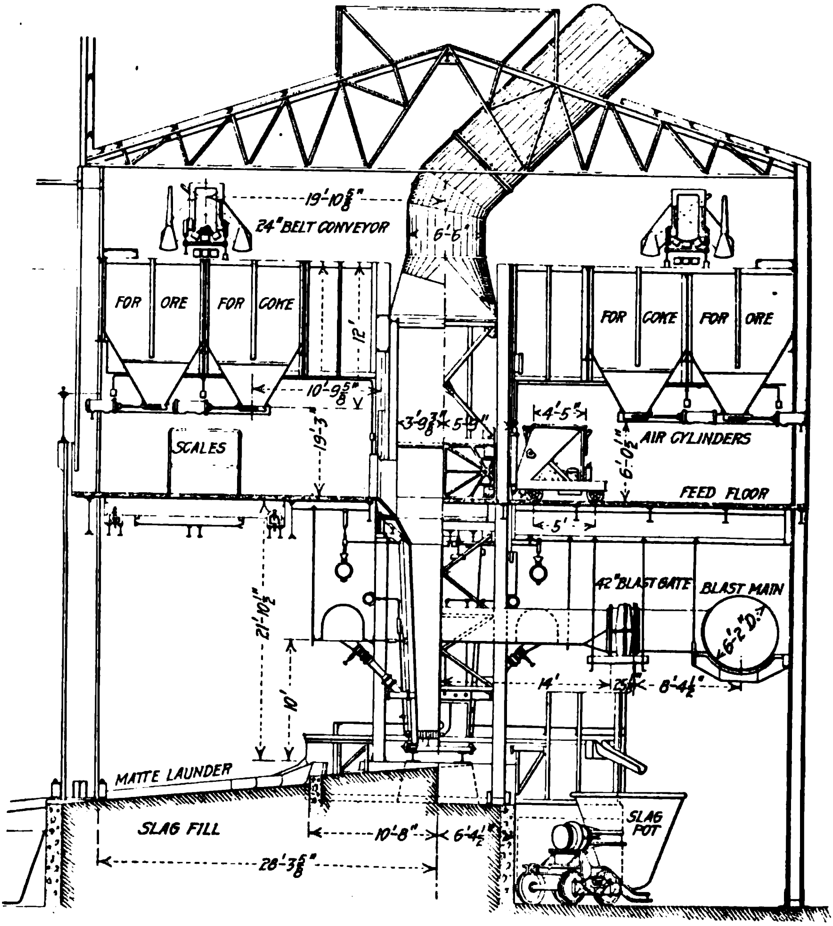


Fig. 9.—End Elevation of Blast-furnace Plant, Calumet & Arizona Mining Co., Douglas.

vided on each side of each furnace. Each furnace has four charge cars, two on each side. The cars are 20 ft. long. When receiving a charge of ore or coke, the cars are resting on track scales. Each car has four compartments and the bins overhead have a gate corresponding to each compartment, so that the amount of charge for any part of the furnace can be regulated to suit the conditions. The car is propelled from the bins to the charging position at the furnace, a distance of 7 ft. or 15 ft., (according to whether a charge of coke or a charge of ore is being moved) by means of an electric motor geared to two wheels of the car. Fig. 10 shows a view of the charge car under the charge bins.

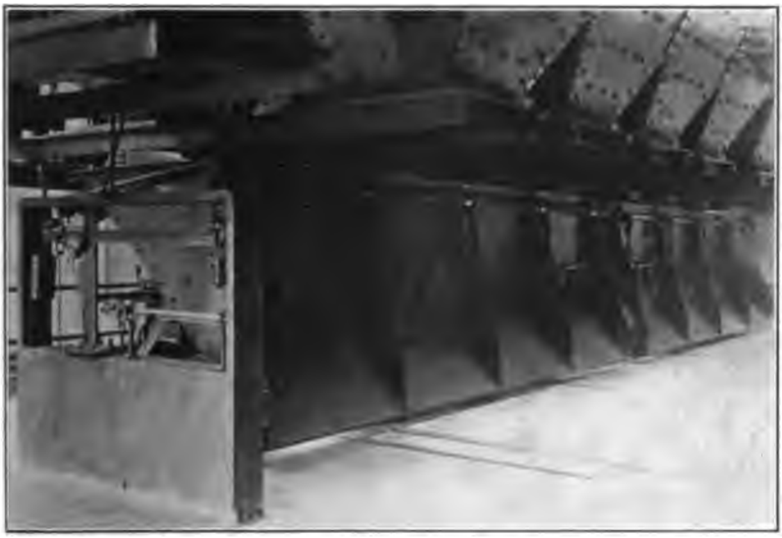


FIG. 10.—VIEW SHOWING CHARGE CAR UNDER BINS.

The furnace side jackets are in one length 16 ft. 6 in. long and are 40 in. wide. It will be noted that the bustle pipe is combined with the truss or girder which takes the outward thrust of the jackets at their midpoint. This simplifies the construction and gives more room around the bustle pipe for the water piping, etc. Fig. 11 is a view showing the tuyères and bustle pipe.

The bridge plates for the furnace bottoms are of structural steel; ½-in. plates are riveted to 6-in. I-beams. The I-beams are spaced 12½ in. apart. The cooling of the bottom is effected entirely by the surrounding air. The I-beams riveted to the plates undoubtedly increase the effectiveness of the cooling. The I-beams supporting the bottom are also used for holding the outward thrust of the bottom ends of the jackets. This thrust is carried into the beams by means of steel rods threaded at one end and flattened at the other. The flattened ends are riveted to the webs of the beams and a nut on the threaded end holds a lug against the bottom end of the jacket.

The furnace tops and charge doors are of structural steel in the form of air jackets. An air space is provided between the inner and outer sheets of the furnace tops. The lower end of this air space connects with the outside air while the upper end connects with the inside of the furnace top. The cool air is drawn in at the bottom and discharged at the top into the furnace gases going to the chimney, thus insuring a positive air circulation which keeps the sheets of the air top properly cooled. The furnace charge doors are jacketed in a similar manner.

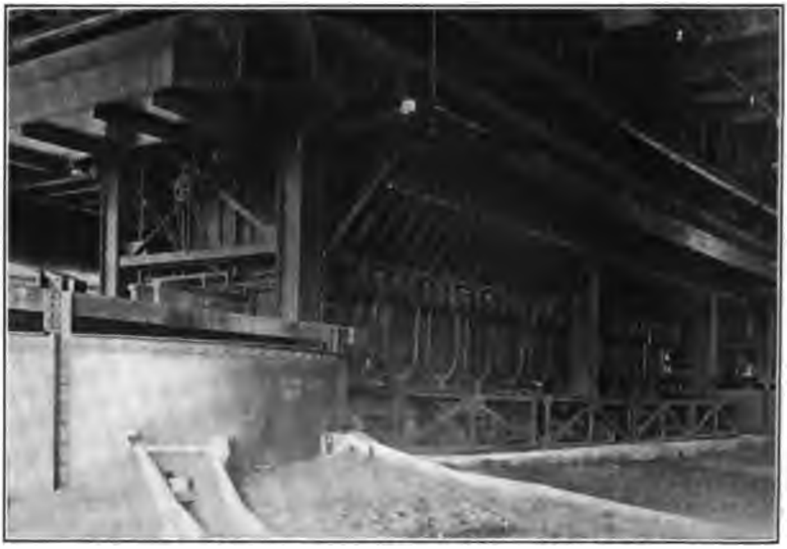


Fig. 11.—View Showing Tuyères and Bustle Pipe, Blast Furnace of Calumet & Arizona Mining Co., Douglas.

These furnaces have been in use for 3 years, smelting as high as 1,400 tons of charge each per day, and no repairs or replacements have been made in furnace water jackets and no warping has taken place in the air-cooled tops.

Reverberatory Furnaces

Fig. 12 shows a side elevation of a typical reverberatory furnace at the International Smelting Co.'s plant which has several features worthy of mention.

(a) *Furnace Bottoms*

It was especially desirable at this plant, on account of difficulties at another plant, to avoid any trouble with furnace bottoms in starting up.

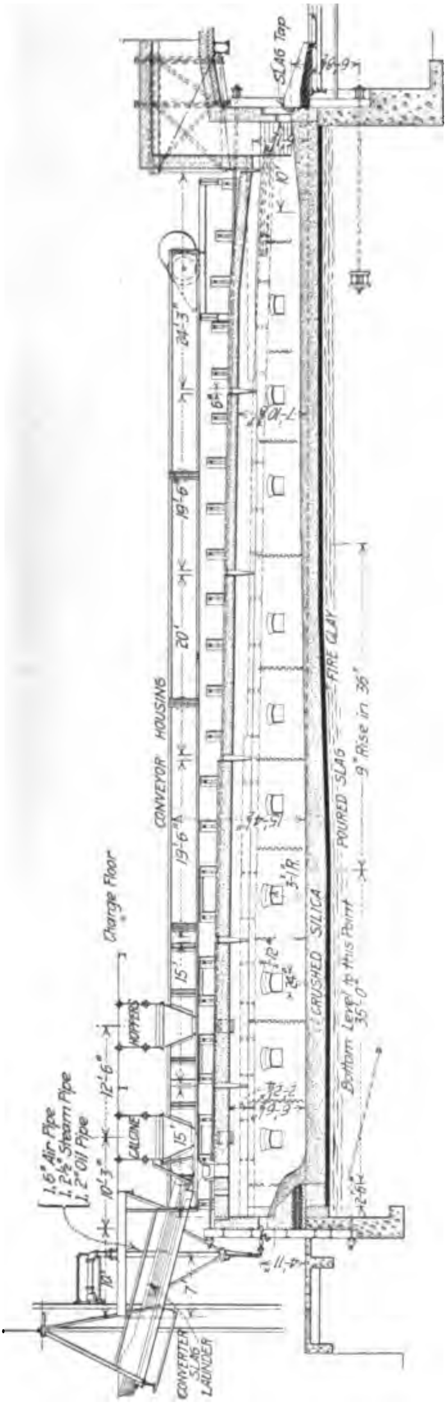


FIG. 12.—LONGITUDINAL SECTION OF REVERBERATORY FURNACE, INTERNATIONAL SMELTING CO.

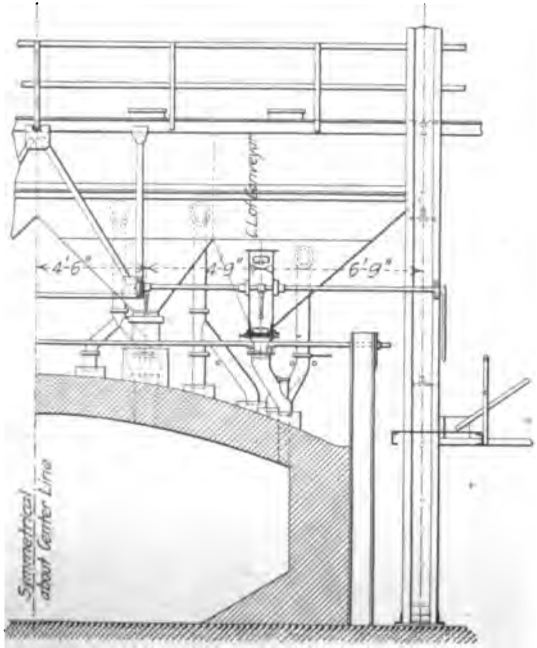


FIG. 13.—SECTION REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

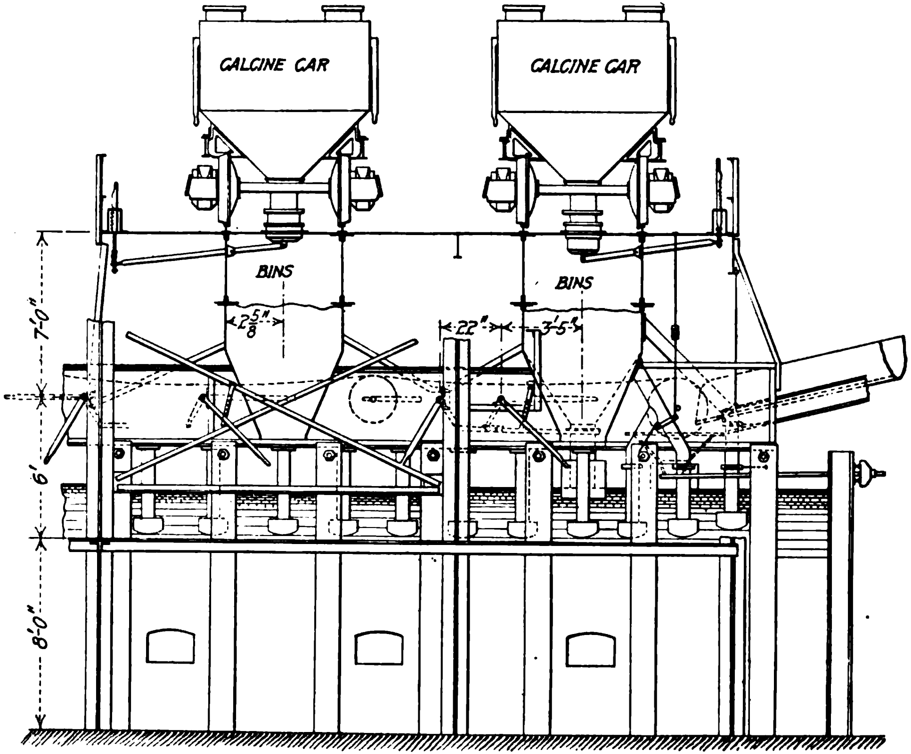


FIG. 14.—SIDE ELEVATION OF REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

To accomplish this, broken slag, brought in railroad cars from the Old Dominion smelter, was melted in a small blast furnace, obtained by setting up a number of discarded furnace jackets and a motor-driven blower, on the site for the reverberatory furnaces. The molten slag thus obtained was conducted in launders to the foundations of the reverberatory furnaces. Heavy concrete beams and struts were provided between the furnaces for taking the thrust from the lower ends of the buck stays. Later, the spaces between the concrete struts and the beams were filled in with molten slag obtained from the regular operation of the reverberatory furnaces. The slag bottom was covered first with a 4-in. layer of fire clay and then a 27-in. layer of silica (94 per cent. SiO_2) crushed to minus ¼ in. These bottoms gave no trouble whatever in starting up.

(b) *Furnace Fettling and Charging System*

The method of charging, instead of fettling only, along the side walls of reverberatory furnaces, now so successfully used by the Canadian Copper Co. and the Anaconda Copper Mining Co., was developed after the construction of this plant was well along. The plant was laid out for charge tracks running at right angles to the furnaces near the firing end. In order to distribute the charge along the sides of the furnaces, charge hoppers were located under the charge tracks directly over the side walls of the furnaces. Drag-chain conveyors were installed, one over each side of each furnace, which received the charge from the charge hoppers (see Figs. 13 and 14). Under these conveyors, approximately every 30 in., suitable down-spouts with gates are provided so that the charge may be distributed along the side walls of the furnaces throughout their length. The bridge wall and side walls at the firing end of the furnaces are charged by drawing directly from the hoppers through suitable spouts.

The floor over the charge tracks around the skimming end of the furnace is paved with firebrick.

(c) *Calcine Cars*

Figs. 15 and 16 show end and side elevations respectively of the calcine car used at the International Smelting Co.'s plant. The feature of this car is the arrangement used for reducing dust losses in receiving and discharging a charge. The car is equipped with four sliding sleeves in the top which are spaced to register with the discharge openings of the calcine hoppers at the roaster plant. When the car is spotted under the roaster hoppers in its proper position, the sleeves are forced up against the calcine hoppers by means of levers on the side of the car. The operating lever is of flat spring steel so that the sleeve can be firmly pressed into con-

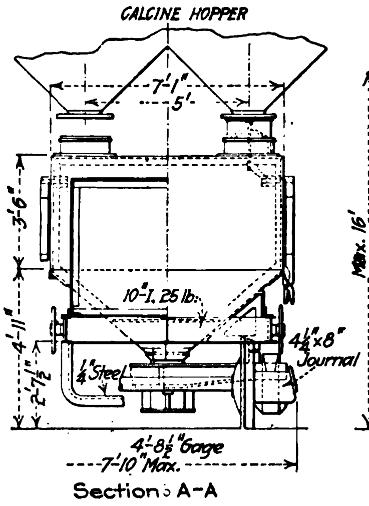


FIG. 15.—END ELEVATION OF CALCINE CAR.

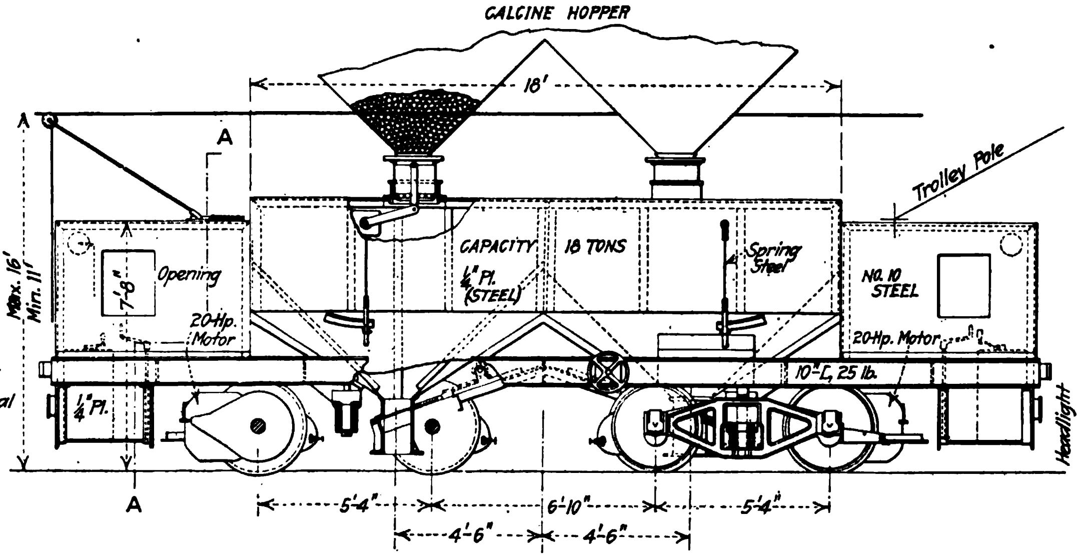


FIG. 16.—SIDE ELEVATION OF CALCINE CAR.

tact with the flange at the bottom of the calcine hopper, and held there by a notch in a quadrant on the side of the car. The spring handle insures a good pressure between the sleeve in the top of the car and the flange of the calcine hopper regardless of the small variations in the distance between the top of the car and the calcine hoppers or any lost motion in the connection of the lever with the sleeve. An air vent is provided in the car made of fine woven wire mesh.

Fig. 14 shows the sleeves in the top of the reverberatory furnace charge hoppers for receiving the charge from the calcine car. These sleeves are similar to the sleeves in the top of the calcine car. By pressing down on the lever, which is operated from the charge floor, the sleeve is forced up against the flange of the discharge spout of the car. Notches are provided in the lever which engage a catch and thus hold the sleeves in their upper position while the car is being discharged. The pressure of the handle is transmitted to the sleeve, as will be seen from the figure, through a helical spring, so that a good contact between the sleeve and the spout of the car is assured regardless of the lost motion and wear in the operating levers and the variations in the distance between the sleeves in the charge floor and the flange on the discharge spout of the car.

(*d*) *Baffling in Waste-Heat Boilers*

By reference to Fig. 17, it will be noted that the header flue between the reverberatory furnaces and the waste-heat boilers, and the connections between the header flue and boilers are well up above the furnace floor level, giving ample head room for the convenience and comfort of the workmen; the bottom of the header flue is 10 ft. 6 in. above the furnace floor level. Also, it will be noted that the boiler-room floor level corresponds to the furnace floor level. This is made possible by reversing the baffling in the Stirling boilers so that the gases enter the front of the boiler near the top and leave the boiler at the rear near the bottom instead of as in the standard setting.

This arrangement of baffling gives equally as good results from the standpoint of water circulation in the boiler, priming, etc., as the standard baffling.

ARRANGEMENT OF CONVERTER PLANTS

Southwestern practice is to have the converter plant adjacent to the reverberatory furnaces, and arranged so that the reverberatory matte is received through launders into ladles directly under the converter plant crane. Also arranged so that the converter slag may be poured from ladles by the converter crane back into the reverberatory furnaces through launders extending from the converter aisle to openings provided in the roofs of the reverberatory furnaces. The converter slag is usually

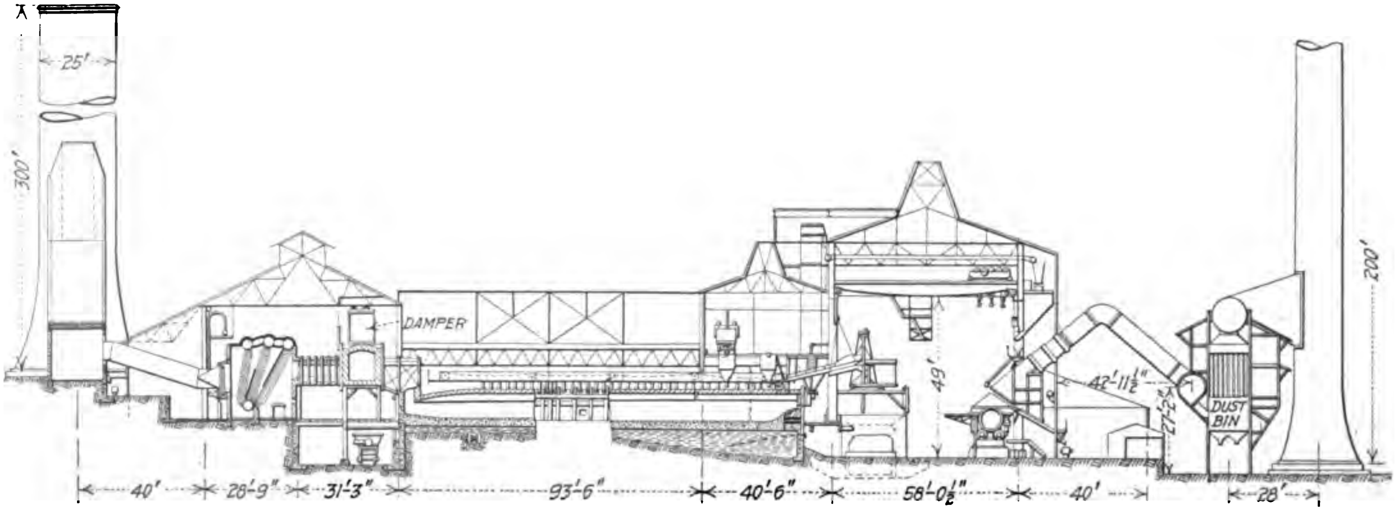


Fig. 17.—Typical Cross-Section of The International Smelting Co.'s Plant.

discharged several feet in front of the bridge-wall of the furnace and midway between the side walls. Fig. 17 is a typical cross-section of the International Smelting Co.'s plant and is typical of the new reverberatory smelting plants in Arizona. Fig. 18 is a plan of the same plant.

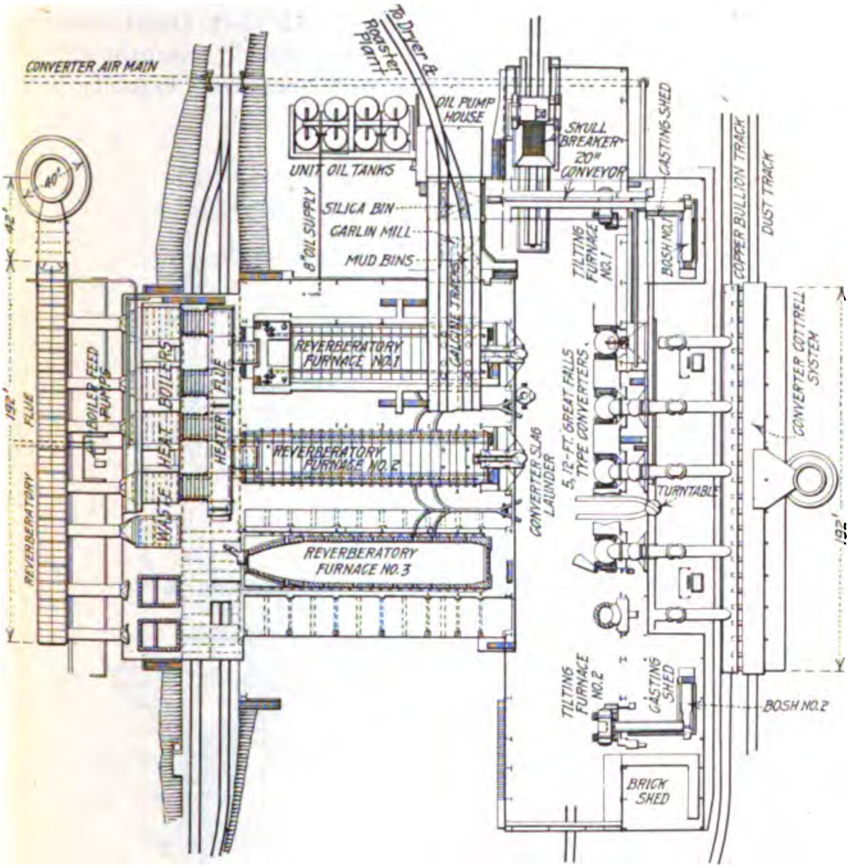


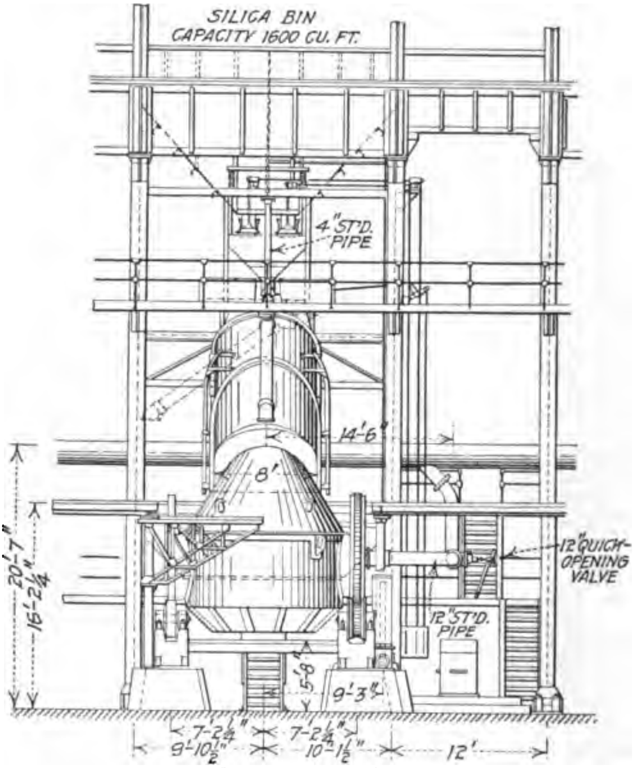
FIG. 18.—PLAN OF INTERNATIONAL SMELTING CO.'S PLANT.

(a) *Siliceous Ore Charging Device for Converter*

Several of the plants are arranged so that the siliceous ore for the converters is drawn directly from bins overhead. Figs. 19 and 20 show a typical arrangement of bin, measuring hopper and chute for accomplishing this operation. This particular arrangement is installed at the International Smelting Co.'s plant. A weighing or measuring hopper is interposed between the overhead bin and the chute leading to the converter mouth. The hopper is mounted on springs and is connected by levers and links to an indicator readily seen from the operating floor. Any desired amount of charge can be weighed out into the measuring hopper and discharged into the converter and this can be accomplished entirely through the operation of levers on the main floor of the converter plant.

(*b*) *Converter Plant Cranes*

Most of the newer plants are equipped with 12-ft. Great Falls type converters. These converters weigh, when lined, approximately 70 tons. Instead of designing the converter cranes for a capacity to lift a converter shell with its lining, as has usually been the practice heretofore, the cranes are designed for a lifting capacity of 40 tons, which is nearer right for the routine work of handling matte, copper and slag in large ladles. Converters with magnesite lining do not have to be moved into and out of their stalls often, but when it is desirable to do this two cranes are used with a lifting rig as shown in Figs. 21 and 22. The converter cranes in all of the Arizona smelting plants are operated by 250-volt direct current. The later cranes are equipped with magnet-switch control.



Front Elevation

FIG. 19.—CONVERTER PLANT, INTERNATIONAL SMELTING CO.

(c) *Skull Breaker for Converter Plant*

Three of the Arizona converter plants are equipped with skull breakers and a fourth plant is now being thus equipped. Figs. 23, 24 and 25 show the general arrangement of the skull breaker at the International Smelting Co.'s plant. A hopper about 11 ft. wide by 24 ft. long is provided, the bottom of which consists of cast-steel bars with projecting lugs, making a grating with openings throughout about 9 in. square. A small traveling bridge is provided over this grating, and mounted on this bridge is a trolley which travels transversely to the path of the bridge. In the trolley is mounted a motor-driven hoist which raises a hammer in the form of a long

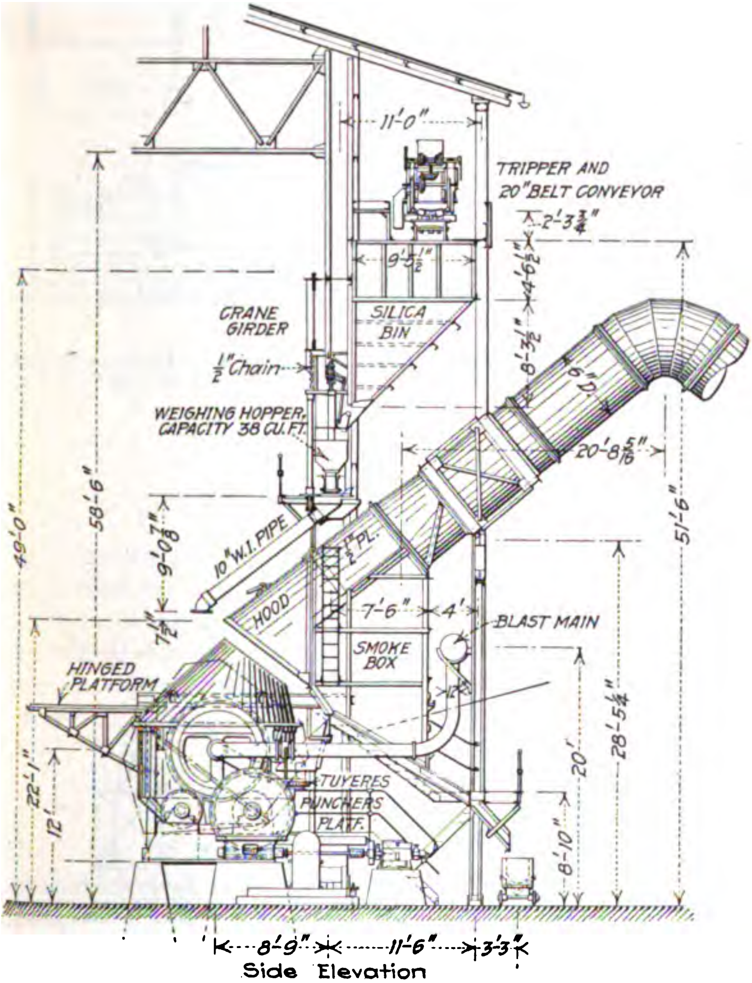


FIG. 20.—CONVERTER PLANT, INTERNATIONAL SMELTING CO.

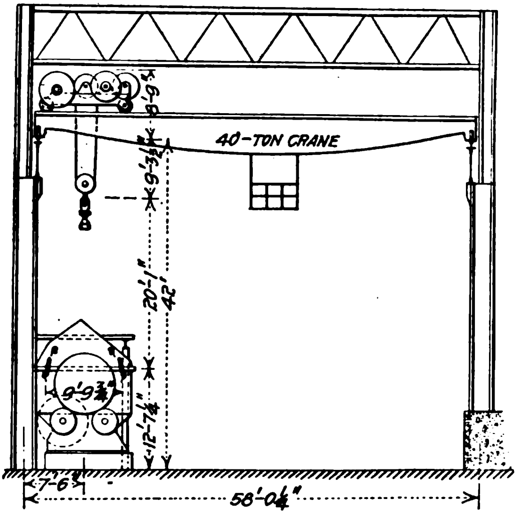


Fig. 21.—Method of Lifting Converters, Calumet & Arizona Mining Co.

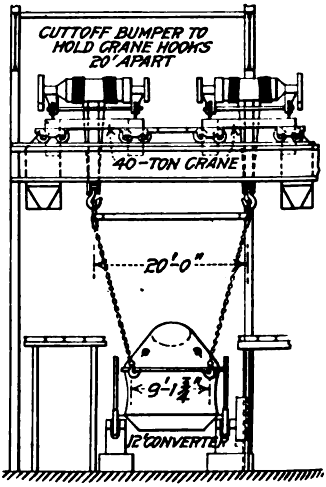
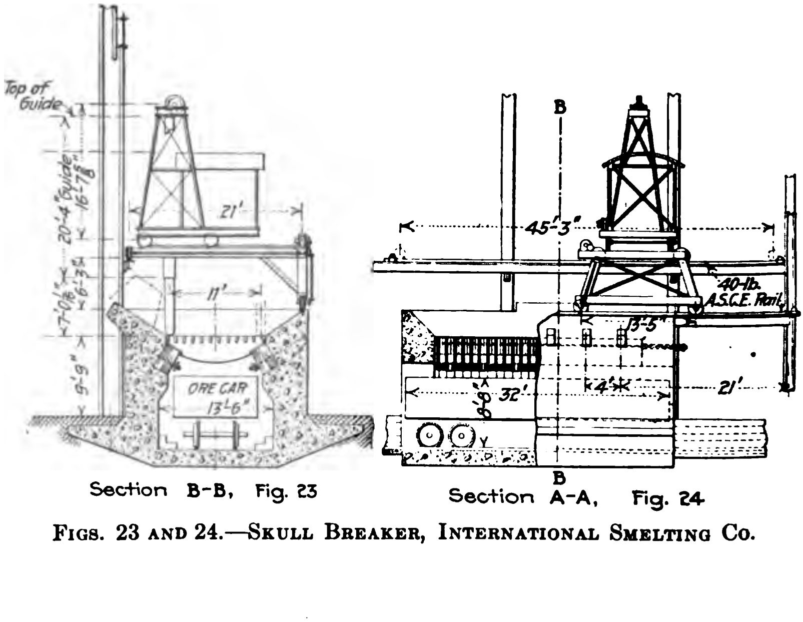


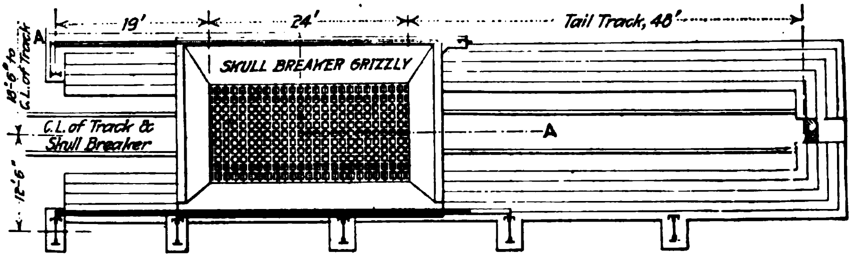
Fig. 22.—Method of Lifting Converters, Calumet & Arizona Mining Co.



Section B-B, Fig. 23 Section A-A, Fig. 24

Figs. 23 and 24.—Skull Breaker, International Smelting Co.

weight that works in a guide. The hammer and guide are mounted on a trolley with the hoist. The hoist operator raises the hammer and allows it to drop on the skull in a manner similar to the operation of the ordinary type of pile-driver hammer. When the skulls are broken into fragments sufficiently small to pass through the grating, they drop into a steel railroad car underneath. The bridge and trolley travel permit the hammer to be spotted over any skull on any part of the grating.



Plan, Fig. 25

FIG. 25.—SKULL BREAKER, INTERNATIONAL SMELTING CO. PLAN.

CHIMNEYS

(a) *The Largest Steel Chimney in the World*

Fig. 26 shows a general elevation with details of the large steel chimney at the plant of the United Verde Copper Co. at Clarkdale, Ariz. It is 30 ft. 9½ in. in diameter inside the steel plates and is 400 ft. 1 in. in height, from the base to the top. It is believed to be the largest steel chimney in existence at this time. It has a brick lining 4 in. thick throughout its height.

(b) *Slag Foundation for Steel Chimney*

Fig. 27 shows the details of the foundation for a steel chimney at the smelting plant of the Calumet & Arizona Mining Co. at Douglas. The chimney is 305 ft. high from the top of the foundation and is 25 ft. 9½ in. in diameter, inside of the steel shell. It has a hollow-tile lining 4 in. thick throughout its height. The feature of the chimney is in the construction of its foundation. The foundation was cast from molten slag hauled to the site for the chimney in the usual slag pots, instead of concrete or masonry as is usual for chimneys of this type. A template for holding the anchor bolts was made of structural angles and channels supported on a central concrete pier and held from turning or moving by a second concrete pier at the outer circumference. The foundation

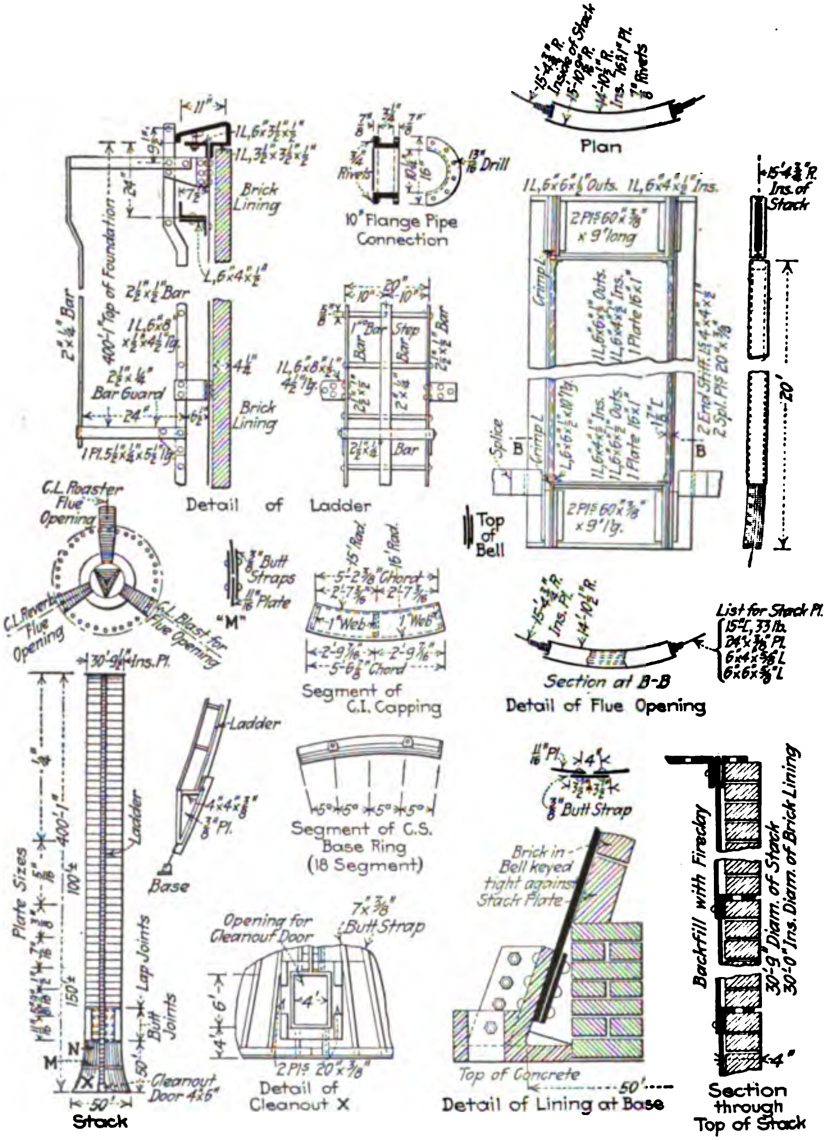


FIG. 26.—GENERAL ELEVATION AND DETAILS OF STEEL CHIMNEY, UNITED VERDE COPPER CO.

bolts were supported at their lower ends on small concrete piers and they in turn supported the template at other points of its outer circumference not supported by the concrete pier, each bolt having at the top a nut on the under side and one on upper side of the template, forming a secure support. Large washers were provided for the bottom ends of the anchor bolts over which were laid old steel rails. Blast-furnace slag was then poured over the foundation and adjacent ground, forming the foundation for the chimney. A concrete capping was laid on top of the slag for the cast-steel base-ring of the chimney proper.

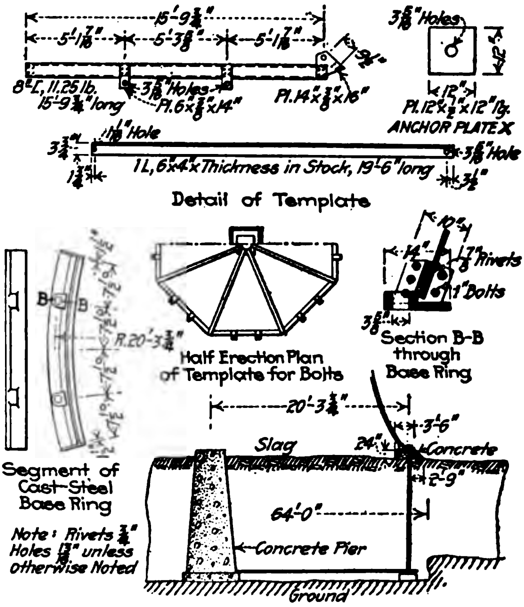


FIG. 27.—DETAILS OF SLAG FOUNDATION FOR CHIMNEY OF CALUMET & ARIZONA MINING CO., DOUGLAS.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The 2,000-Ton Leaching Plant at Anaconda

BY FREDERICK LAIST,* B. S., AND HAROLD W. ALDRICH,† M. E., ANACONDA, MONT.

(Arizona Meeting, September, 1916)

AFTER a series of experiments covering a period of about three years, extending from the spring of 1912 to the spring of 1915, a 2,000-ton leaching plant for the treatment of the accumulated copper concentrator tailing was built and put into operation. During the experimental period, the first step was that of laboratory experiments or beaker leaches. The results on this small scale were so satisfactory that a small operating plant, capable of handling 5 tons of roasted tailing per day, was installed. Again, the results proved satisfactory and an 80-ton plant was built and operated continuously from August until February, 1913. In this plant, full-sized roasting and leaching units were

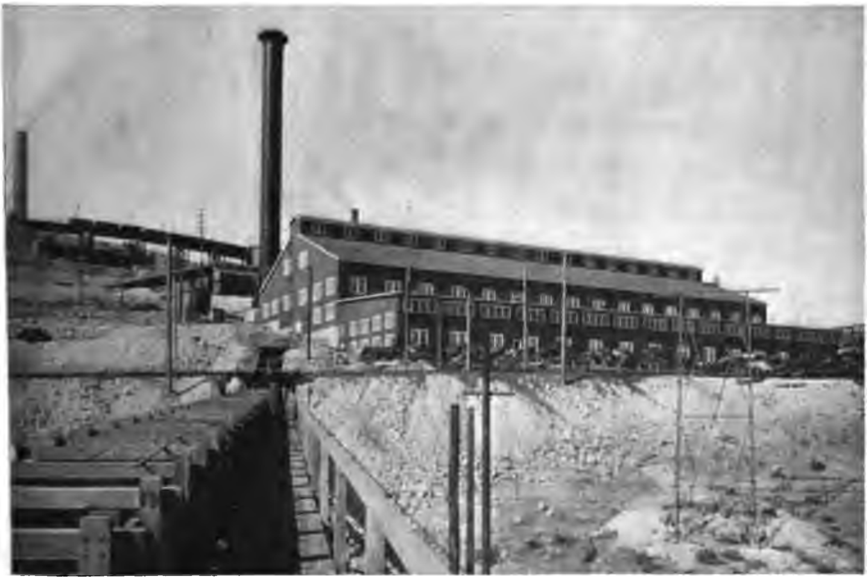

FIG. 1.—LEACHING PLANT AT ANACONDA.

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* Metallurgical Manager, Anaconda Mining Co.

† Superintendent of Leaching Plant, Anaconda Mining Co.

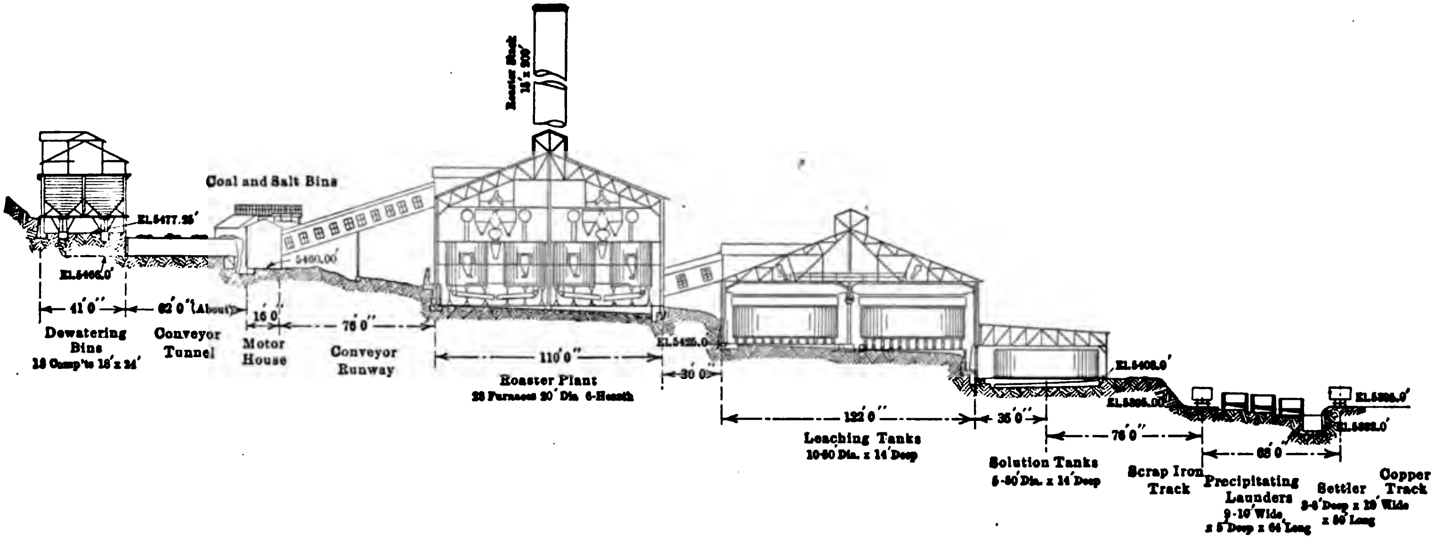
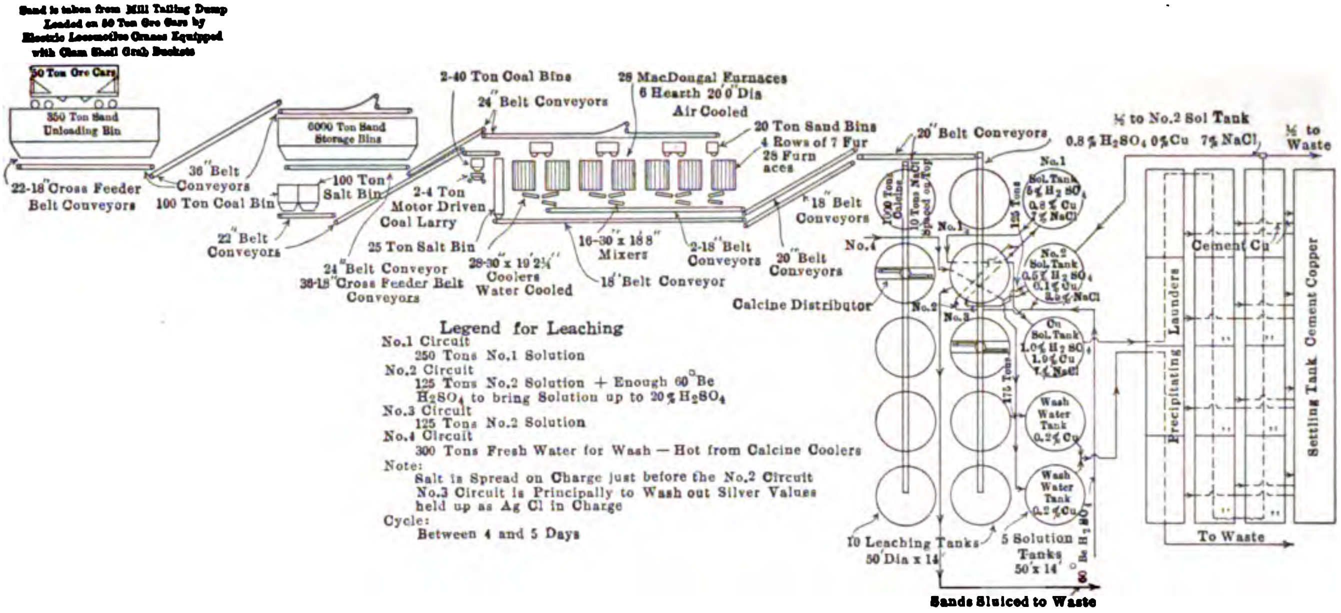


FIG. 2.—GENERAL PROFILE OF 2,000-TON LEACHING PLANT AT ANACONDA.


Sand is taken from Mill Tailing Dump Loaded on 50 Ton Ore Cars by Electric Locomotive Cranes Equipped with Clam Shell Grab Buckets
50 Ton Ore Cars
350 Ton Sand Unloading Bin
22-18" Cross Feeder Belt Conveyors
36" Belt Conveyors
100 Ton Coal Bin
6000 Ton Sand Storage Bins
100 Ton Salt Bin
22" Belt Conveyors
24" Belt Conveyor
38-18" Cross Feeder Belt Conveyors
2-40 Ton Coal Bins
24" Belt Conveyors
2-4 Ton Motor Driven Coal Larry
25 Ton Salt Bin
28-30" x 19' 2½" Coolers Water Cooled
28 MacDougal Furnaces 6 Hearth 20' 0" Dia Air Cooled
16-30" x 18' 8" Mixers
18" Belt Conveyor
2-18" Belt Conveyors
20 Ton Sand Bins 4 Rows of 7 Furnaces
18" Belt Conveyors
20" Belt Conveyors
Calcine Distributor
20" Belt Conveyors
100 Tons Calcine
10 Tons NaCl Spread on Top
No.1
No.4
No.2
No.3
125 Tons
175 Tons
No.1 Sol. Tank 5% H₂SO₄ 0.8% Cu 7% NaCl
No.2 Sol. Tank 0.5% H₂SO₄ 0.1% Cu 2.5% NaCl
Cu Sol. Tank 1.0% H₂SO₄ 1.9% Cu 7% NaCl
Wash Water Tank 0.2% Cu
Wash Water Tank 0.2% Cu
60° Be H₂SO₄
10 Leaching Tanks 50' Dia x 14'
5 Solution Tanks 50' x 14'
Sands Sluiced to Waste
½ to No.2 Sol Tank 0.8% H₂SO₄ 0% Cu 7% NaCl
½ to Waste
Precipitating Launders
Cement Cu
Settling Tank Cement Copper
To Waste

Legend for Leaching

No.1 Circuit
250 Tons No.1 Solution
No.2 Circuit
125 Tons No.2 Solution + Enough 60° Be H_2SO_4 to bring Solution up to 20% H_2SO_4
No.3 Circuit
125 Tons No.2 Solution
No.4 Circuit
300 Tons Fresh Water for Wash — Hot from Calcine Coolers

Note:
Salt is Spread on Charge just before the No.2 Circuit
No.3 Circuit is Principally to Wash out Silver Values held up as Ag Cl in Charge

Cycle:
Between 4 and 5 Days

FIG. 3.—FLOW SHEET OF LEACHING PLANT.

used. The results obtained by the operation of the 80-ton plant proved that the roasting and leaching of deslimed concentrator tailing could be profitably done, so in the early spring of 1914, the construction of a 2,000-ton leaching plant was begun (Figs. 1 and 2). Operation of the plant began on May 13, 1915. A flow sheet of the plant is shown in Fig. 3.

Leaching Plant Feed

The accumulated tailing in the New Works dump is estimated at about 20,000,000 tons. The dump consists of the concentrator tailing discharge, over a period extending from February, 1902, to the present date. According to daily samples taken during that period, the copper content is about 0.64 per cent. and the silver 0.48 oz. per ton. The peak of the dump, or the point where the tailing is discharged, carries about 0.75 per cent. copper, while down toward the toe, and where present excavation is taking place, the copper content is only 0.57 per cent. About 3 lb. per ton of the copper is oxidized, the remainder being sulphide.

The following shows an average screen analysis of the tailing in the dump:

Cumulative Per Cent.

+20.0	+40.0	+60.0	+80.0	+100.0	+160.0	+200.0	−200.0
22.6	64.7	82.0	89.1	93.8	97.7	98.5	1.5

An average analysis of tailing being treated at the present time follows:

H_2O Per Cent.	Cu Per Cent.	Ag Ounces	SiO_2 Per Cent.	FeO Per Cent.	S Per Cent.	Al_2O_3 Per Cent.	CaO Per Cent.
5.63	0.573	0.487	81.0	2.8	1.9	9.8	0.4

The dump is being excavated by a Bay City Industrial Works electric hoist, equipped with a 4-yd. bucket. The hoist loads the tailing into 50-ton ore cars. These are hauled to the leaching plant storage bins in trains of 15 cars each. The hoist is capable of hauling 3,000 tons in 8 hr.

Conveying and Storage Equipment

Unloading Pit

The loaded cars of tailing are spotted by means of a 25-ton electric locomotive operated by the third-rail system. The unloading pit is a steel bin capable of holding 350 tons and is of sufficient length to allow three cars to be unloaded at a time. On top of the pit is a 2-in. grizzly, through which all tailing must pass before going to the storage bins. This grizzly serves to keep out rocks or lumps which would block the feeders, furnaces, etc. Under the steel bin are 22 short-belt feeders, each feeding from its own gate and running at right angles to the length of the bin. These feeders discharge onto a 36-in. belt running lengthwise, which in turn delivers the sand to another 36-in. belt traveling on an 18° angle to the top of the storage bins. The system will handle over 3,000 tons of tailing in 8 hr.

Storage Bins

The storage bins will hold 6,000 tons of sand tailing. This gives between two and three days' supply for the leaching plant, in case of railroad troubles or difficulties due to cold weather. The bins are of substantial wood construction and are inclosed, the walls and roof of the building consisting of wood sheathing covered with corrugated iron. The bins are arranged in a double row and are hopper-bottomed. The tailing is distributed over the bins by means of a 36-in. belt and movable tripper.

Underneath are 36 gates, each with its hopper from which an 18-in. belt feeder delivers to a 24-in. belt running the full length of the building, in the center. By means of other belts, traveling through a tunnel under the railroad tracks, and up an 18° incline, the feed is delivered to the top of the furnace building.

There are also small coal and salt storage bins from which a 22-in. belt system conveys coal and salt to bins in the roaster building, the salt being drawn from there to the leaching building as it is required. All belts run at approximately 400 ft. per minute.

ROASTING EQUIPMENT

The furnace building is 232 by 110 ft. and is of steel and concrete construction (Fig. 4).

There are 28 McDougall-type, six-hearth furnaces arranged in four rows of seven each. The furnaces are 20 ft. in diameter, each being equipped with two fireboxes, diametrically opposite, the flame entering over a fire bridge directly into the third hearth, the top hearth being designated as the first. The grate dimensions of each firebox are 3 by 4 ft. (Figs. 5 and 6).

Each furnace has a 20-ton feed hopper, to which the tailing is delivered by means of two 24-in. belts each equipped with a movable tripper. The furnaces are fed by 14-in. belt feeders drawing from these hoppers, the amount of feed being controlled by gates which are operated by means of a screw adjustment, the feed dropping through a hole in the top arch, directly onto the top floor (Fig. 7). During the time the feed is on the two upper floors, it is dried and heated; as it drops

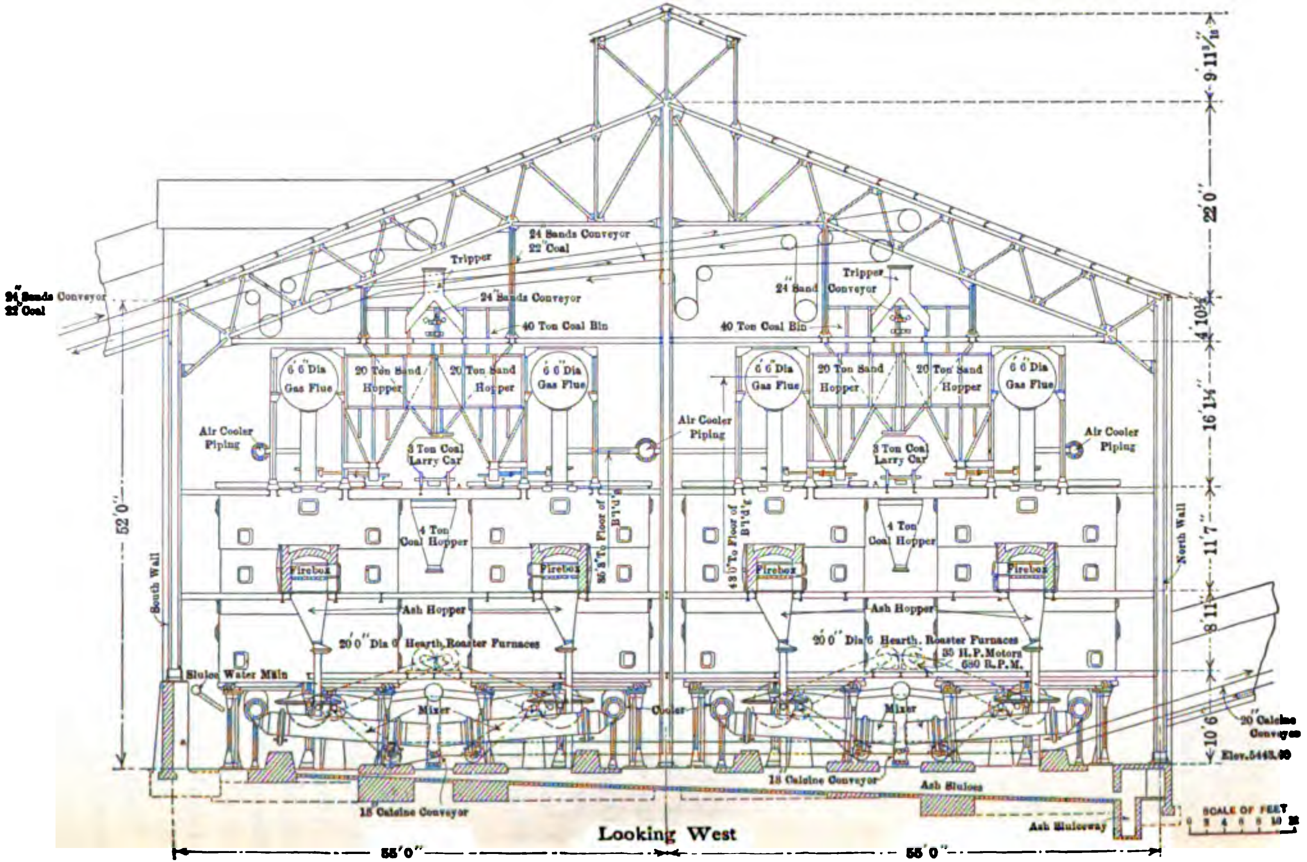


FIG. 4.—CROSS-SECTION THROUGH ROASTING PLANT.

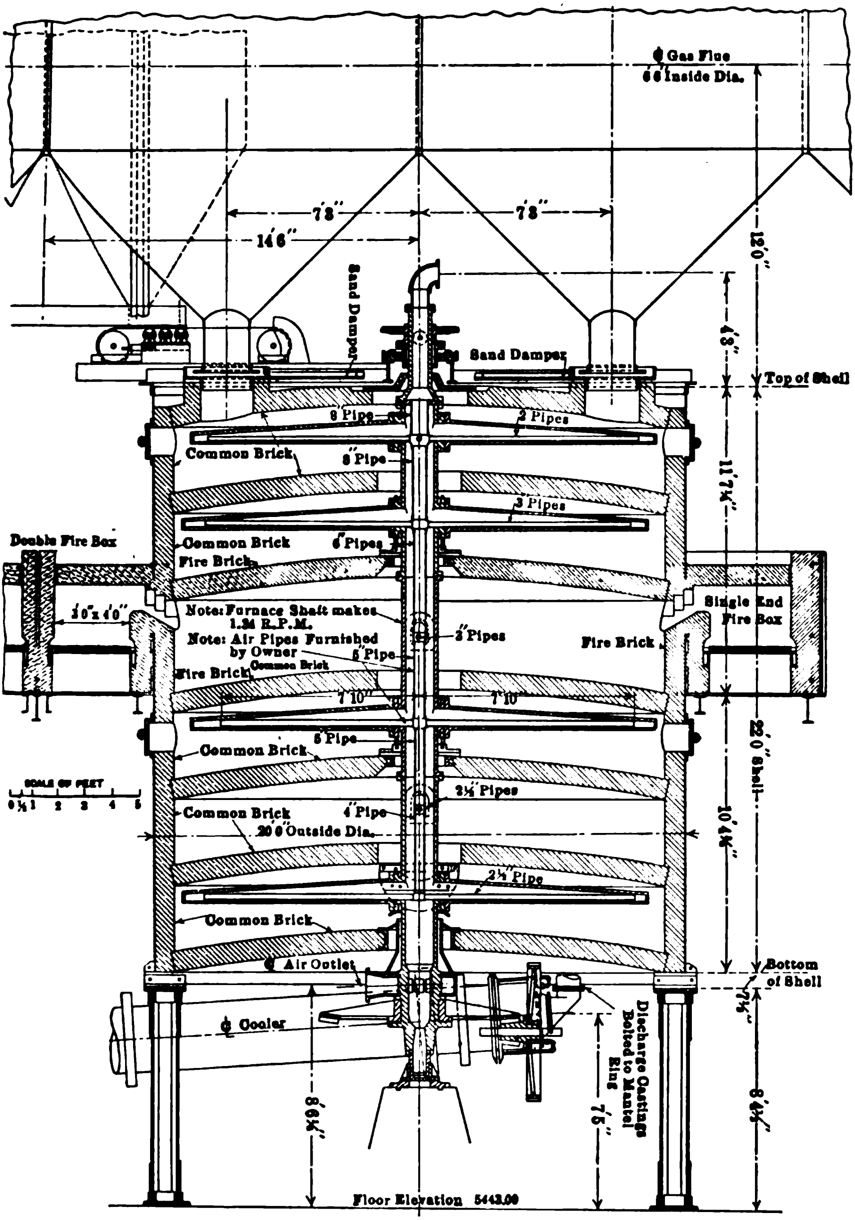


FIG. 5.—VERTICAL SECTION THROUGH McDOUGALL FURNACE.

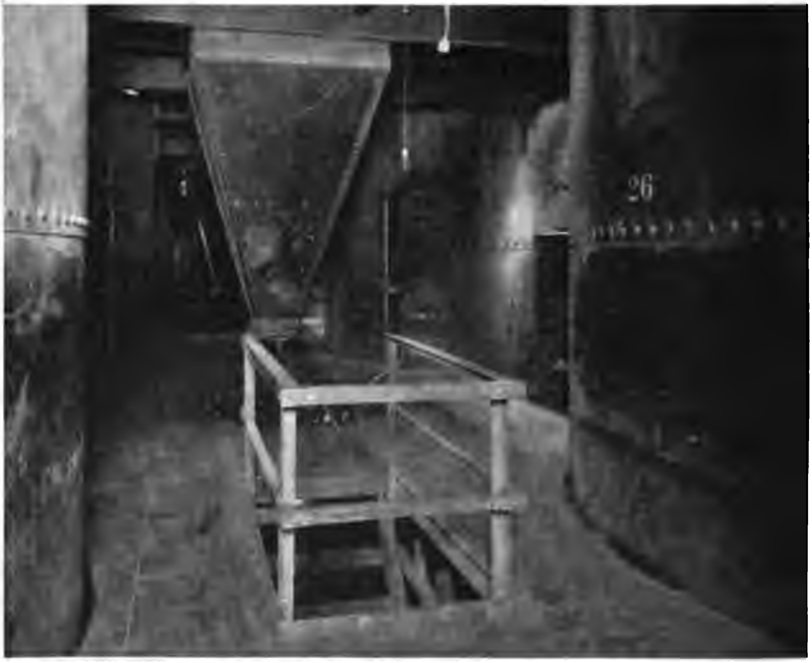


FIG. 6.—FIRING FLOOR OF ROASTER PLANT.

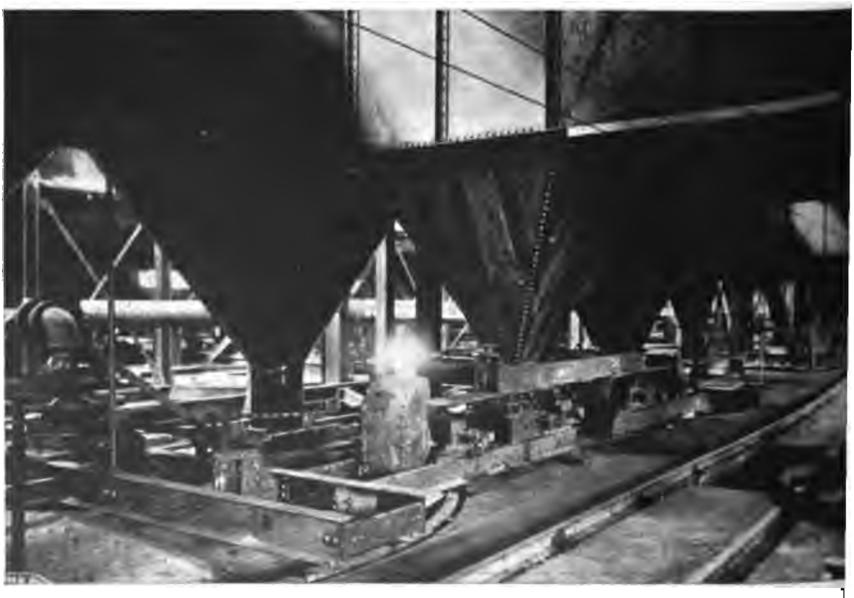


FIG. 7.—FEED FLOOR OF ROASTER SHOWING FEED HOPPERS, FLUE CONNECTIO AND FEEDING MECHANISM.

to the third or fired floor, the sulphur ignites. The three lower floors are kept hot by the combustion of the sulphur in the tailing.

Four flues run the length of the building, one over each row of seven furnaces. Each furnace has two opposite connections, from the top hearth to the flue. All four flues lead into a balloon flue with a downtake of 45°. The balloon flue enters the 15 by 200-ft. steel stack with a 45° uptake. The stack is unlined. In the bottom of the balloon flue is a 6-in. screw conveyor which delivers the flue dust to the belt-conveyor system which receives the calcine from the furnaces.

The furnaces are air-cooled, the air being furnished by four No. 11 Buffalo blowers each direct-connected to a 50-hp. motor. The air intake is at the top of the furnace shaft and the discharge at the bottom. The hot air does not enter the furnaces, but is delivered to the leaching and solution buildings for heating purposes by a suitable piping system. When it is not needed for this it is discharged into the atmosphere.

Each furnace is equipped with a cylindrical cooler, 30 in. in diameter and 19 ft. long. The cooled calcine enters a mixer or concrete-lined steel cylinder, at the head end of which a very small stream of water is added to settle the dust. The mixer discharges a moist warm calcine to an 18-in. conveyor belt, and by a system of conveyors the calcine is delivered to the leaching building.

The ashes from the fireboxes drop into launders and are sluiced out through the main tailrace.

LEACHING PLANT EQUIPMENT

The leaching building is 293 by 122 ft. and is of steel and wood construction. It contains 10 redwood tanks each 50 ft. in diameter, and 14 ft. deep. The average charge to a tank is about 1,000 tons of calcine.

The tanks are equipped with an ordinary filter bottom, made of 1¼-in. slats resting on 2 by 4-in. pieces. Above this are two layers of heavy cocoa matting and on top of the matting is a grating, made of 1¾ by 3½-in. material, with 6-in. square spaces. The grating fills with calcine 3½-in. deep and serves to keep the force of the sluicing water from tearing the matting. The acid solutions rot the cocoa matting, but if not disturbed, it will hold its shape and be an efficient filtering medium long after it is too much decomposed to handle. All the steel tank hoops are covered with lead pipes to protect them against leaks of acid solutions.

The tanks are in two rows of five each. A 20-in. conveyor belt travels over each row, and by means of a tripper, the calcine is dumped into a suitable distributor, which spreads it over the tank (Fig. 8).

Each tank has three lead pipes 4 in. in diameter and one 4-in. iron pipe entering at the top. The lead connections are for strong and dilute acid solutions and the iron pipe is for wash water. Above the level of the leaching tanks an iron storage tank is provided for holding the stock of concentrated acid. Its capacity is about 120 tons of 60° Bé. acid. All concentrated acid, used to raise the acid strength of any solution, is added to the solution as it goes on the charge in the leaching tanks.

There are seven 10-in. sluicing gates in the bottom of each leaching tank, one in the center and six spaced equidistant from each other in a circle about half way between the center and the circumference of the tank. These discharge into launders which connect with the main tailings launder.

"Acimet" valves and lead piping are used throughout for handling dilute and concentrated acid solutions.

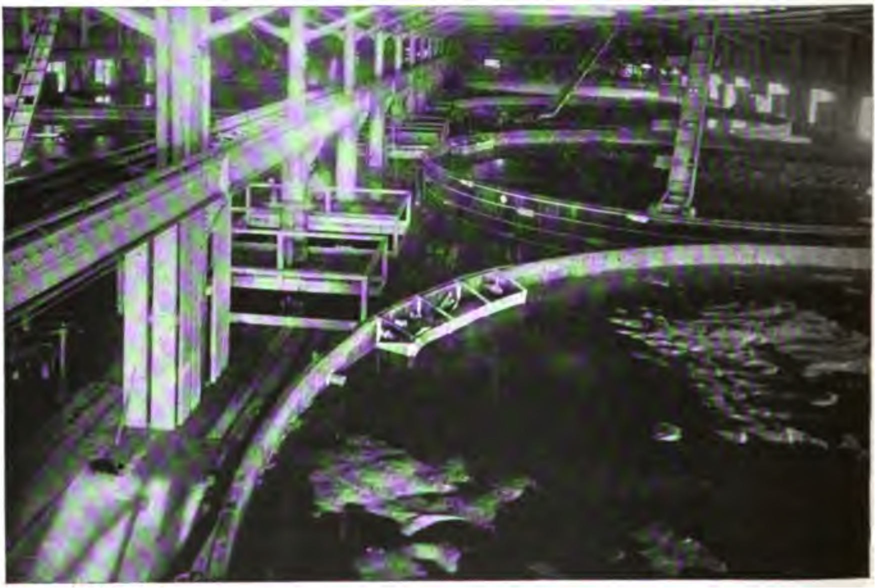


FIG. 8.—VIEW OF LEACHING TANKS AND DISTRIBUTING CHUTES.

The floors of both the leaching and solution buildings are of concrete, and are painted with an asphalt-tar mixture for acid proofing. These floors slope to a gutter which connects to a pump sump and in this way any overflow or leakage of solution is saved and returned to the system.

The solution tank building is a leanto off the leaching building and contains five solution-storage tanks. These are 50 ft. in diameter and 14 ft. deep. Solutions drain from the leaching tanks to the storage tanks and are pumped to the top of the leaching tanks, from the solution tanks, by means of vertical shaft, direct-connected, hard-lead, centrifugal pumps.

PRECIPITATING DIVISION

The precipitation of the copper and silver is accomplished with scrap iron. The precipitating launders are of concrete, each about 250 ft. long and, having a section of 4 by 8 ft. available for containing iron. Each launder is partitioned off into four sections by concrete walls. Any of the 12 sections may be bypassed for the purpose of cleaning up. In the bottom of the launders is a heavy wood grating, upon which the iron rests, leaving a space about 6 in. under it, for accumulation of any cement copper which may drop off the iron. In the side of each section, at the bottom, are four 6-in. holes, toward which the concrete bottom slopes. These holes discharge into launders which carry the copper to a settling tank. There it is washed and stored and finally excavated with a clamshell bucket, loaded into standard railroad cars and, at present, shipped to the briquetting plant before blast-furnace treatment. An electrically operated Brown hoist, equipped with a lifting magnet, handles the scrap iron and loads the copper.

DETAILS OF OPERATION

Roasting

The tailing is subjected to a simple oxidizing roast, no particular care being taken to obtain a large amount of sulphate. The sulphur content of the feed is about 2.2 per cent. and that of the calcine about 0.6 per cent. One-third of the total sulphur in the calcine is in the form of sulphate. When too hot a roast is attempted in order to decrease the total sulphur content, a certain portion of the copper is rendered insoluble in all ordinary acids, with the exception of hydrofluoric.

A pyrometer is inserted over the fourth floor of every furnace, and by means of these, the firemen are able to keep constant control of the temperature. The best results are obtained by keeping the fourth floor at about 500° C. The hottest hearth in the furnace is the third or fired floor, and averages about 535° C.

The water for the calcine cooler system enters at about 40° C., is discharged at 65° C., and is piped to the solution building, where it is used to heat the circulating solutions. It is then pumped back and used again in the coolers. The calcine after passing through the coolers, and after the addition of 1 per cent. moisture, while going through the mixers, has a temperature of about 45° C. During the conveying from the roasters to the leaching tanks, this temperature is lowered to 40° C.

Leaching

The leaching is done by continuous downward percolation, no circulation or upward percolation being used. The percolation rate will vary from 3 in. per hour with the first solution to as high as 10 in. per hour with the wash water. As nearly as possible, all solutions and wash waters go on the charge at 40° to 50° C.

It requires about one-fourth of the weight of calcine, in weight of solution, to saturate a charge thoroughly.

There are five solution tanks: One for storage of No. 1 solution, one for No. 2 solution, one for copper solution, and two for wash-water.

	Cu Per Cent.	H_2SO_4 Per Cent.	NaCl Per Cent.
No. 1 solution tank	0.8	5.0	7.0
No. 2 solution tank	0.1	0.5	3.5
Copper-solution tank	1.9	1.0	7.0
Wash-water tanks	0.2	1.0	1.0

After a tank is charged with calcine and leveled, 250 tons of No. 1 solution is added as fast as the charge will absorb it. The drain valve is always open, so, as soon as the solution reaches the bottom of the tank, it commences to drain to the copper-solution storage tank as copper solution. From the copper-solution tank there is only one outlet, which is to the precipitation launders. After traveling through the launders, two-thirds of the solution is returned to the No. 2 solution tank and the balance wasted. This waste is necessary to keep impurities such as iron and aluminum sulphates from building up in the system.

When the No. 1 solution has all been added to the leaching tank, the solution is allowed to drain until none shows on top of the calcine, when 1 per cent. of the weight of the charge, of common salt (NaCl), is spread over the calcine. On top of the salt is then added 100 tons of solution from No. 2 solution tank, but with additional strong acid to bring it to 20 per cent. H_2SO_4. Following the 20 per cent. acid solution, 150 tons of No. 2 solution is added, but without additional strong H_2SO_4. This scheme gives a zone 4 or 5 ft. in depth of very strong chloridizing solution, traveling down through the charge. There is about 8 per cent. of ferrous and ferric iron in solution, which, with the salt, forms ferric chloride, in itself a very corrosive reagent, even dissolving a considerable amount of unroasted sulphide. This chloridizing action also gives the silver extraction, as without it very little silver is recovered. The 150 tons of No. 2 solution which follows the 20 per cent. acid is for the purpose of washing out silver chloride and dissolved copper which may have been held in the calcine. It carries very little copper or acid, but is fairly high in salt content, and therefore better than a clean water wash. Following the last acid solution, about 300 tons of hot, clean water is added.

The two portions of No. 2 solution, one at 20 per cent. acid, and the other at 0.5 per cent. acid, after percolating through the charge, drain to the No. 1 solution tank.

The wash water, less a quantity sufficient to make up for the discarded solution, drains to the two wash-water tanks. The balance goes to the No. 2 solution tank and adds enough to make up the amount of solution discarded from the precipitating division each day.

Precipitating

The practice here is too old to necessitate much explanation. The main advantages in the practice at this plant over the usual practice are the large launders, which make it possible to put in large and odd-shaped pieces of iron, and the presence of salt in the solutions which prevents the copper from plating on the iron, and makes a soft spongy cement copper which is easily washed off with a hose, leaving the iron clean for more precipitation. It is never necessary to remove the iron for cleaning. The silver is recovered by precipitation on the precipitated copper.

RESULTS

The resulting cement copper carries about 70 per cent. copper. The following data are taken from the reports for the month of October, 1915. This is a representative month, but it is certain that the results will be improved upon, after longer operation.

Sand tailing treated, tons	70,401.00
Copper in feed, per cent	0.575
Silver in feed, ounces per ton	0.45
Copper in tailing, per cent	0.082
Silver in tailing, ounces per ton	0.14
Sulphuric acid (60° Bé.), pounds per ton of feed	64.90
Coal, per cent. of feed	3.30
Salt, per cent. of feed	1.52
Iron, pounds per pound of copper	2.00

The plant during this month made an extraction of about 80 per cent. of the copper and 60 per cent. of the silver. This is less than indicated by assays of heads and tailings owing to various plant losses of which the largest is dust from the roasting furnaces amounting to about 4.5 per cent. of the copper in the feed.

Analyses of Feed and Tailing

	Cu, Per Cent.	Ag, Ounces	SiO_2, Per Cent.	FeO, Per Cent.	S, Per Cent.	Al_2O_3 Per Cent.	CaO, Per Cent.
Feed	0.575	0.45	81.3	3.0	2.1	9.4	0.4
Tailing	0.082	0.14	84.7	2.4	0.4	8.7	0.4

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Mining and Milling Practice at Santa Gertrudis

BY HUGH ROSE,[*] PACHUCA, MEXICO

(Arizona Meeting, September, 1916)

THE properties of the company lie within the Pachuca district, State of Hidalgo, Mexico, connected by three railway lines with Mexico City, 55 miles southwest, and by two lines with Vera Cruz, 250 miles southeast.

The ores were formerly divided by sorting into two classes, smelting and milling, the former averaging about 2 oz. gold and 335 oz. silver, the latter 0.12 oz. gold and 23 oz. silver. The smelting ores were sold to custom plants, principally the American Smelting & Refining Co. at Aguascalientes. The milling ores were treated by the patio process at the Guadalupe Hacienda, at Pachuca. This patio was probably the largest in existence at the time and continued in active operation up to March, 1910.

In January, 1910, the mines and patio were sold to English interests, represented by Camp Bird Limited. Two new companies were formed, the Compañia de Santa Gertrudis, S. A., to operate the mines, and the Compañia Beneficiadora de Pachuca, S. A., to build and operate a custom cyanide milling plant. (Fig. 1.)

GEOLOGY

The historical geology of the Pachuca district is described by Fred J. Pope as follows:

1. The most recent sedimentary rocks in the district belong to the Cretaceous period.

2. The Cretaceous rocks were penetrated by large intrusions of andesite (referred to hereafter as primary andesite) which occur as elliptical dome-shaped masses, striking northwest-southeast. These andesite bodies are from 8 to 15 miles long and from 6 to 10 miles wide.

3. A period of erosion during which the contour of the andesite domes was changed.

4. Andesite flows (secondary andesite) which rest unconformably on the primary andesites.

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[*] General Manager, Compañia de Santa Gertrudis, S. A.

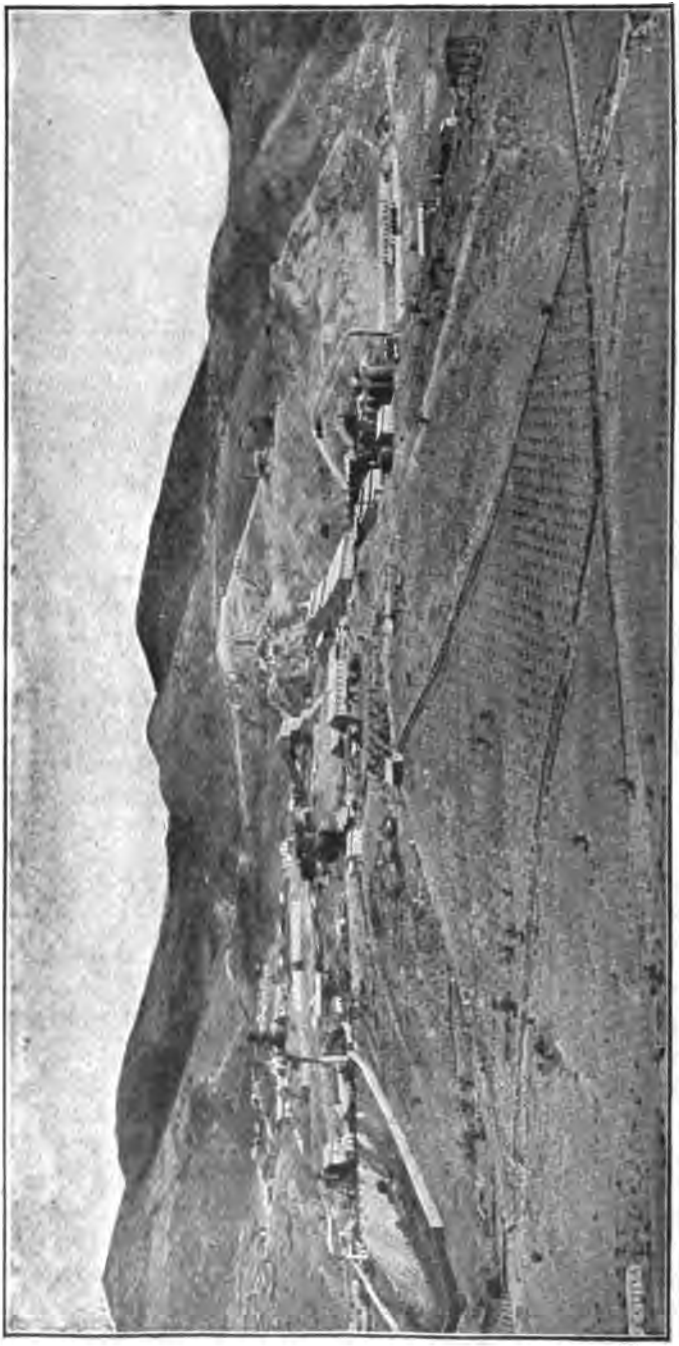

Fig. 1.—Santa Gertrudis Mine and Mill.

5. A second period of erosion, again exposing areas of primary andesites.

6. A series of rhyolite flows which partially covered the primary and secondary andesites.

7. A third period of erosion which exposed large areas of both the primary and the secondary andesite. During this period there were many intrusions of quartz-porphyry dikes which cut the primary and secondary andesites and the rhyolite. For the most part they do not affect the economic tenor of the veins. In occasional instances, however, the veins are locally impoverished where they cut these dikes. Relatively to the extent of the orebodies, these dikes are very small and practically have no economic bearing.

8. A period of areal faulting, probably due to subsidence and adjustment caused by the large, dome-shaped masses of primary andesite cooling at depth. During this period fissures were formed, rock between the walls was brecciated and there was a heavy silicification, but practically no economic values were brought into the veins. It was essentially a period of fissuring, brecciation and silicification.

9. A second period of faulting with movements relatively much less than during the first period. It was during and immediately subsequent to this secondary faulting that the veins received their economic values.

10. Erosion and minor faults.

Country Rock

In discussing the magmatic variations of the primary andesite, Mr. Pope further states:

The primary andesite in which the Santa Gertrudis vein occurs is not a homogeneous rock. There are local variations due to differentiation in cooling. For the most part the andesite has a slight grayish-green color, is porphyritic, and breaks with a rough surface. This is the prevailing normal type and averages approximately 65 per cent. silica. A more basic variation averages a little over 60 per cent. silica, is porphyritic, dark green in color, breaks with a conchoidal fracture and is very tough. Between these two types there are all gradations and one often graduates imperceptibly into the other. Both occur at all levels to the bottom of the mine, a depth of 2,000 ft.

Operations in the Pachuca district have demonstrated that the tough basic phase of the primary andesite is not favorable to the formation of ore and that all the large orebodies are found in the normal type. This is due to the fact that after the primary faulting and silicification, the basic phase was as tough or tougher than the original rock and thus resisted the secondary fracturing and subsequent introduction of economic values.[1]

Nature and Formation of the Veins

The veins are not simple fissures filled with silica and economic minerals, but in a general way may be described as crushed zones varying from a few feet up to 50 ft. and in some instances over 100 ft. in width. Usually both foot and hanging walls are fairly well defined, but in many places the dividing line between ore and wall rock can be determined only by systematic sampling.

As a rule the hanging wall is heavy, due to fracturing and to slips or faults parallel to the vein, and in consequence must be carefully supported when mining the ore.

Ore Occurrence

The veins of the district are notable for the size and persistence of the oreshoots.

As a rule, the oreshoots do not outcrop but are discovered at varying depths of one hundred to several hundred feet from the surface, and in the upper levels are shorter and often have a distinctive pitch, generally to the east. Usually, several of these shoots combine in depth to form the main great oreshoot of the mine. In the Santa Gertrudis vein a number of the smaller shoots were thus worked to a depth of from 800 to 900 ft. At about this horizon they joined together forming an orebody which persisted for another 600 to 700 ft. in depth, having a length in the several adjoining properties of about 3,000 ft. and a width averaging 18 ft. The grade of this great oreshoot was not less than $20 per ton.[2]

At a horizon of about 1,800 ft. from the surface the veins, though still strong, become rapidly impoverished and at 2,000 ft. little payable ore is to be found. Below this horizon explorations are being conducted to discover the possible existence of a new zone of mineralization, but such work is not sufficiently advanced to permit any forecast of results.

The ores generally are clean, carrying very small quantities of lead, zinc or iron, and little or no manganese, all of which are unimportant economically. The principal economic mineral is silver sulphide—argentite. The commercial values are confined to gold and silver, which are present in the proportion of 15 per cent. and 85 per cent. respectively. By weight, for each 1,000 oz. of silver there are usually 5 oz. of gold present.

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[1] Mr. Pope's report was made after a brief examination undertaken at the time of the purchase of the properties by the Camp Bird interests. Subsequent developments over 5 years have confirmed his findings in general. Several ore-bearing veins occur in basic andesite areas but oreshoots are small and of relatively small economic importance. In the main North Vein, wherein the principal oreshoots are located, the lean parts of the vein within the ore-bearing horizon usually occur where the vein passes through the tough basic phase.

[2] U. S. currency and dry short tons used throughout unless otherwise specified.

MINING SYSTEMS AND METHODS

The mining systems in general use are:

(a) Overhead stoping without timbering, with filling.
(b) Overhead stoping with timbering and filling.
(c) Overhead stoping with shrinkage; with and without subsequent filling.

In mining virgin ore in place, system (a) is most generally adopted, being used where the hanging wall is too heavy to employ shrinkage stoping, but sufficiently strong to hold until the stope can be filled. Fig. 2 illustrates the details of this system. Foot-wall drifts (counter-drives) in the harder andesite are kept open after the sill floor of the vein has been stoped out and filled. Crosscuts from the counter drive are run through the vein every 165 ft. and heavily timbered with chutes for drawing the ore from the stopes. These chutes are carried up with cribbed timbers as the stope advances and are divided into two compartments, one for ore and one for manway. Between each two crosscuts a raise is put up to provide entrance for the waste used in filling. Spreading of this waste is done with wheelbarrows.

Throughout the Pachuca district the re-mining of the old areas has become of great importance. The old fills are generally payable and in most cases ore in place is also found on the walls. At first sight it might appear that such territory would be cheaper to mine than virgin ground. But this is not the case, for both the hanging wall and the ore are heavier, making it necessary to adopt system (b) with resulting slower extraction of ore at higher cost. Fig. 3 illustrates the details of this system which in the main follow those of system (a) with the addition of square-set timbering. As the timbers are soon covered by the waste filling, light sets of 8 in. square material or of equivalent round posts are sufficient. Sets measure 7 ft. 5 in. high by 5 ft. by 5 ft. in the clear.

System (c) is applicable only in one small stope in the Santa Gertrudis mine, and hence need not be discussed in detail. In general it is used more commonly in the Real del Monte district than on the Pachuca side, the walls in the former district being much harder and more secure.

Levels were formerly driven 75 to 100 ft. apart. More recently the distance has been increased to 165 ft. (50 m.) vertically. It is questionable whether a greater distance than this could be used economically because of difficulty of holding open the timbered chutes.

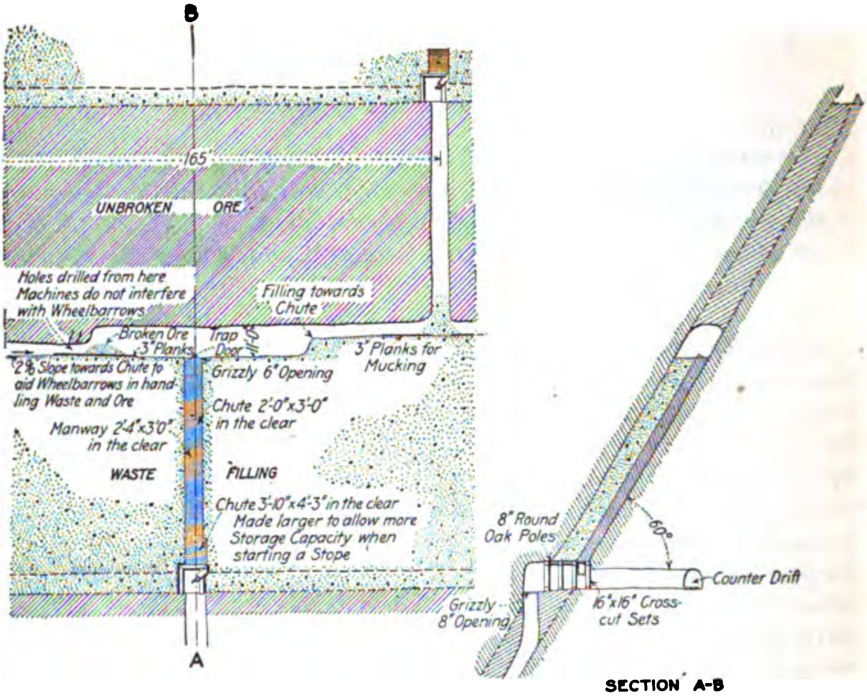

B
165'
UNBROKEN ORE
Holes drilled from here Machines do not interfere with Wheelbarrows
Filling towards Chute
Broken Ore
3" Planks
Trap Door
3" Planks for Mucking
2% Slope towards Chute to aid Wheelbarrows in handling Waste and Ore
Grizzly 6" Opening
Chute 2'-0"x3'-0" in the clear
Manway 2'-4"x3'-0" in the clear
WASTE
FILLING
Chute 3'-10"x4'-3" in the clear Made larger to allow more Storage Capacity when starting a Stope
A
8" Round Oak Poles
60°
Counter Drift
Grizzly 8" Opening
16"x16" Cross-cut Sets
SECTION A-B

Fig. 2.—Overhand Stoping without Timbering and with Filling

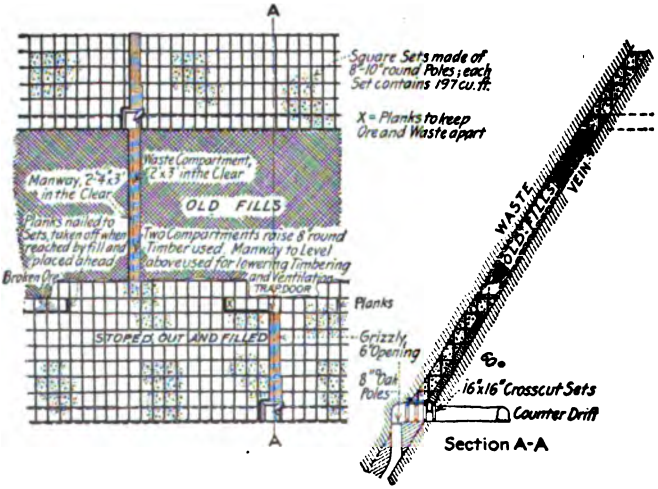

A
Square Sets made of 8"-10" round Poles; each Set contains 197 cu. ft.
X = Planks to keep Ore and Waste apart
Waste Compartment, 2'x3' in the Clear
Manway, 2'-4"x3' in the Clear
OLD FILLS
Planks nailed to Sets, taken off when reached by fill and placed ahead
Two Compartments raise 8" round Timber used. Manway to Level above used for lowering Timbering and Ventilation
Broken Ore
TRAP DOOR
Planks
STOPED OUT AND FILLED
Grizzly, 6" Opening
8" Oak Poles
A
WASTE
OLD FILLS
VEIN
60°
16"x16" Crosscut Sets
Counter Drift
Section A-A

Fig. 3.—Overhand Stoping with Timbering and Filling.

Ore Breaking

Ore breaking is entirely overhand, using air-hammer drills and ⅞ in. 40 per cent. gelatine powder. Filling for the stopes formerly obtained from caving stations in the hanging wall, is now derived partly from development headings. To provide excess requirements a quarry is being opened on the surface. A central raise in hard foot-wall rock is being put up to tap this supply. This raise will deliver or receive waste at each level.

Development

Shaft sinking is generally done against considerable water, 200 to 400 gal. per minute; 3⅛-in. machine drills are used. Electric firing of rounds has been tried, but with only partial success, due to the breaking timbers which must be carried close to the bottom. Electric time fuses have been used with better success and when intelligently handled are an improvement upon ordinary fuse firing.

Of special interest was the construction of a two-compartment shaft (San Francisco shaft No. 2) adjacent to the existing three-compartment San Francisco shaft No. 1. A pillar of rock, 20 ft. wide, was left to protect the latter shaft, which was kept in operation; sinking and raising were carried on coincidently from several different levels. The shaft raises were timbered in two compartments, a light cage being used in the manway to handle men and tools. A small electric hoist at the foot of the raise served the cage, the hoisting rope being carried over a sheave swung from a movable beam at the top of the raise timbers. Sinking was done by usual methods.

Work was commenced in March, 1912, and the shaft completed to a depth of 1,768 ft. from the collar to the 19th level, with pockets at a number of levels, and put into operation in May, 1913. Steel timbering, bought cut to proper lengths but fabricated locally, was installed throughout. No. 18 galvanized corrugated sheet lagging was used, but only where necessary, being slipped into place on the outside of sets and held secure by small steel clips, riveted to the sheets and fitted over the flange of the I-beam wall plates. The cost of the shaft without pockets was:

	Per Foot
Raising and sinking	$24.62
Timbering (steel)	20.33
Loading and tramming	0.65
Hoisting (waste to surface)	6.23
Explosives	2.92
Lighting	0.59
Miscellaneous	0.27
	$55.61

All drifting and crosscutting of importance is done with 3⅛-in. machine drills, using the horizontal bar mounting, two machines to a bar, the muckers cleaning up the breakage while the top part of the round is being drilled.

Raising is entirely accomplished with air-hammer drills.

Winzing is generally by hand and only a relatively small amount of it is done, raising being generally cheaper and more rapid. Air-hammer drills are occasionally used in winzing in hard ground.

Average Development Costs per Foot

	Shaft Sinking (against Water)	Drifting and Crosscutting	Raising	Winzing
Labor	$59.25	$3.75	$2.50	$4.75
Timber	8.45	0.10	0.35	0.10
Explosives	3.25	1.70	0.70	0.60
Lighting	0.70	0.05	0.06	0.06
Tracking and piping	1.00	0.60	0.12	
Power	15.60	0.50	0.50	0.55
Maintenance, machine drills	0.50	0.05	0.10	
Tools	0.45	0.10	0.10	0.10
Tool sharpening	1.50	0.60	0.60	0.10
Hoses, pump fittings, etc	1.50	0.05	0.05	
Lubricants	0.25	0.02	0.02	
Supervision	1.05	0.30	0.30	0.30
Totals	$93.50	$7.82	$5.40	$6.56

Tramming

Tramming underground is by hand and electric haulage. Four main levels are equipped with electric haulage using 2.5-ton Jeffrey 250-volt trolley locomotives. A battery locomotive was tried without success, the equipment being insufficient to handle the load against varying grades and track curvatures. In the locomotive haulage, a train is made up of 10 U-dump cars each of 20 cu. ft. capacity, equal to about 0.9 dry ton of Santa Gertrudis broken ore. The cars are equipped with Whitney roller-bearing wheels and, in general, are giving good satisfaction. The center of gravity is sufficiently low to require complete filling for easy dumping and to allow the cars to be emptied over the shaft pockets without stopping the train. Waste, when required, for stope filling, is loaded into the empty cars from the waste chutes on the return trip. On hand-tramming levels, the ore is passed, wherever possible, to electric-haulage levels for transportation to shafts. Tracks are 20 in. gage, with 16- and 20-lb. rails.

Hoisting

Men and material are handled in San Francisco shaft No. 1, by two double-deck cages operating in and out of balance. These are served by an Allis Chalmers double-reduction geared electric hoist, using a three-phase, 50-cycle, 440-volt, 300-hp. induction motor with grid type resistance, at a rope speed of 1,000 ft. per minute; also in San Juan shaft, two single-deck cages operating in balance and served by a converted 150-hp. electric hoist, with a rope speed of 700 ft. per minute.

Ore and waste are handled in San Francisco shaft No. 2 by two 5-ton skips operating in or out of balance. These are served by a Fraser and Chalmers single-reduction, herringbone-geared hoist using a three-phase, 50-cycle, 440-volt, 350-hp. induction motor with liquid controller type resistance, at a rope speed of 1,000 ft. per minute, and a capacity of 800 tons per 8-hr. shift; also in San Guillermo shaft, one 3.5-ton skip operating counterbalanced and served by a duplicate of the hoist used on San Francisco No. 1, the rope speed being 1,000 ft. per minute and the capacity 300 tons per 8-hr. shift. This compound shaft is of interest because of its unusual characteristics. It is of old-style masonry from collar to 9th level. It is heavily timbered through the vein to the 13th level and has a twist of 30° between the 13th and 15th levels to bring it parallel to the vein. Between the 15th and 16th levels a 50-ft. radius compound converts the shaft to 60° incline at which it continues to the 20th level, a total vertical and inclined length of 2,090 ft. Except in the heavy part through the vein, little trouble has been experienced in the operation of this shaft, despite its small size in the upper section.

Fig. 4 shows details of construction of San Francisco No. 2 skip, of special interest being the trunnion wheels, the duplicates of which have given no trouble in use in San Guillermo compound shaft. These wheels are also utilized in dumping the skip, as shown in Fig. 5. The loop in the dumping path engages the bottom wheel, thereby preventing the skip from falling back into the shaft in case the safety detaching hook should not hold during an overwind. This dumping path is designed to take care of a skip for vertical or inclined service.

Fig. 6 shows electric-hoist brake-release contact brought into play by being struck by the ascending bale of the skip cage. The hoist brakes, air-operated, are under control of a solenoid connected to the brake weights. When the electric contact is made, the solenoid drops these weights independently of the engineer, thereby applying the brakes.

Plough-steel ropes of flattened-strand construction are used, 1 in. in diameter on men and materials hoists and 1⅛ in. diameter on the ore hoists. Ropes are systematically inspected, following South African requirements. After 2 years' service, new ropes are put on, even though the old ropes show no undue wear.

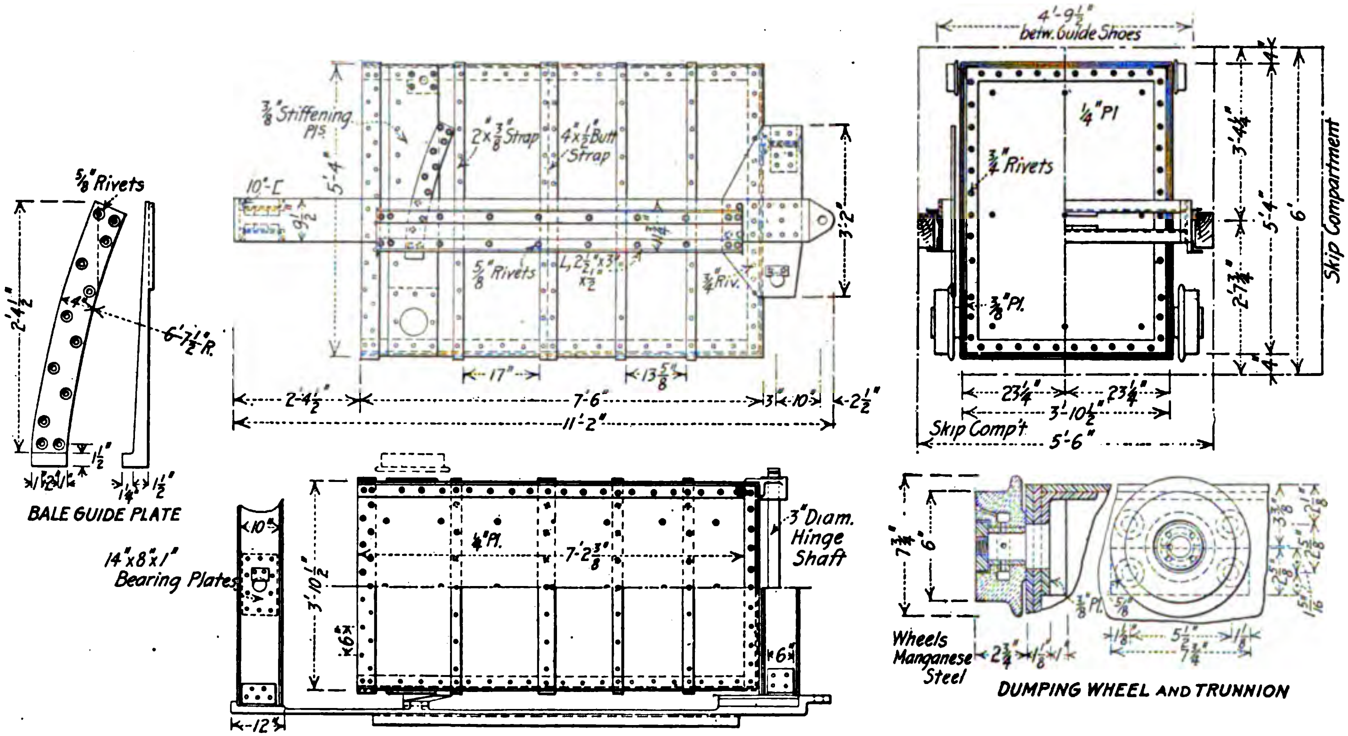

5/8" Rivets
2'-4 1/2"
4"
6'-7 1/2" R.
1 1/2"
1 1/2"
1 1/4"
BALE GUIDE PLATE
14"x8"x1" Bearing Plates
10"
12"
3/8" Stiffening Pls.
10"-[
9 1/2"
5'-4"
2"x3/8" Strap
4"x1/2" Butt Strap
5/8" Rivets
L 2 1/2"x3"x1/2"
3/4" Riv.
3'-2"
17"
13 5/8"
2'-4 1/2"
7'-6"
3"
10"
2 1/2"
11'-2"
3'-10 1/2"
6"
1/4" Pl.
7'-2 3/8"
3" Diam. Hinge Shaft
6"
4'-9 1/2" betw. Guide Shoes
1/4" Pl
3/4" Rivets
3/8" Pl.
4"
3'-4 1/4"
2'-7 3/4"
5'-4"
6'
Skip Compartment
23 1/4"
23 1/4"
3'-10 1/2"
Skip Comp't. 5'-6"
7 3/4"
6"
3/8" Pl.
5/8"
Wheels Manganese Steel
2 3/4"
1 1/8"
1 1/8"
5 1/2"
7 3/4"
1 1/8"
DUMPING WHEEL AND TRUNNION

FIG. 4.—SKIP

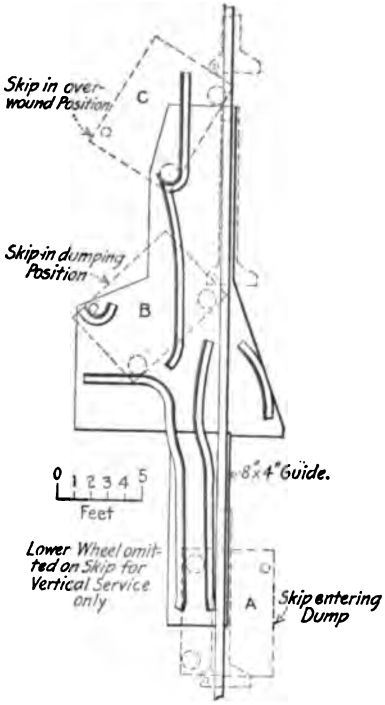

Skip in over-wound Position
C
Skip in dumping Position
B
0 1 2 3 4 5
Feet
8"x4" Guide.
Lower Wheel omitted on Skip for Vertical Service only
A
Skip entering Dump

FIG. 5.—DUMPING PATH IN HEADFRAME.

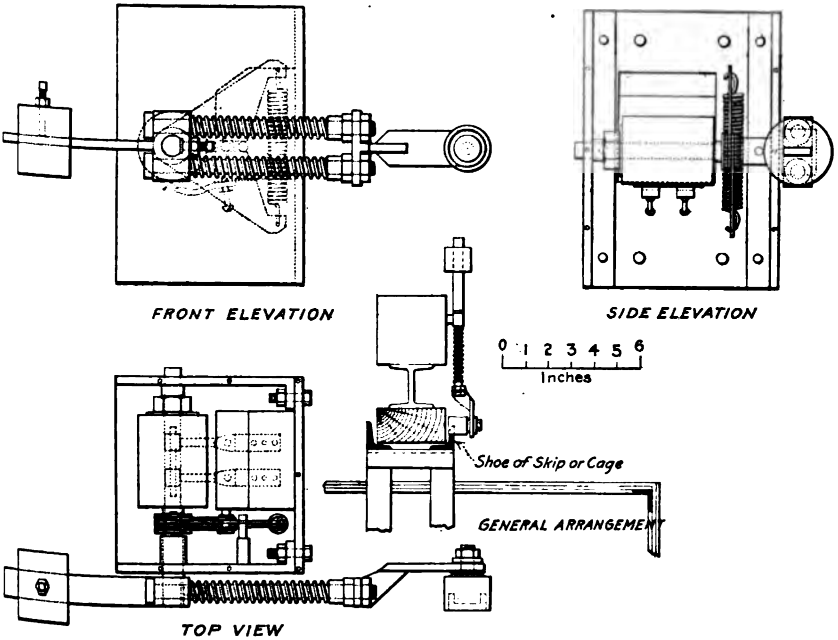
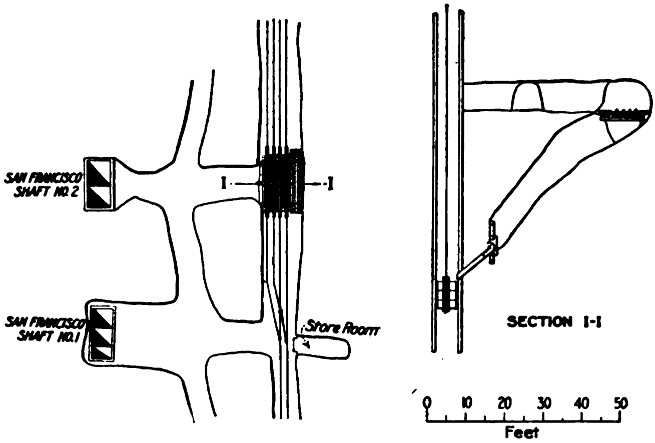


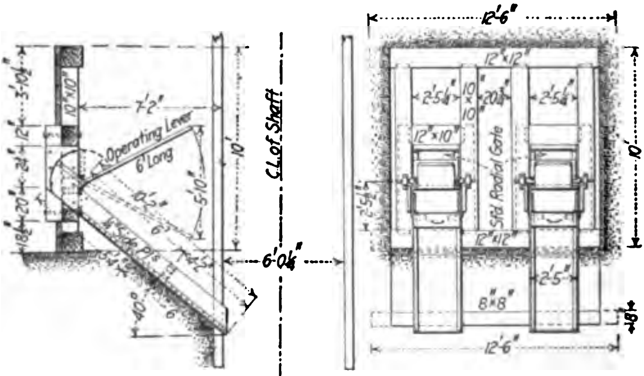
FIG. 6.—ELECTRIC-HOIST BRAKE-RELEASE CONTACT.

Pockets serving the skips are located on all working levels. Fig. 7 shows the characteristic arrangement of the grizzly on top of the pocket and the tracking layout; Fig. 8 shows the standard loading station in general use. Because the ore is free-running and unusually fine, no measuring chutes are required, skip loading being accomplished accurately and with minimum loss of time by manipulation of the gates. With ample ore supply, *i.e.*, pocket fairly full, the loading time is from 15 to 20 sec.



PLAN

FIG. 7.—STATION AND POCKET LAYOUT.



Section

Front Elevation

FIG. 8.—SHAFT-LOADING STATION.

Signals from cage or skip tender to hoist engineer are by electric bells, the same system being used by persons calling at any level. Bells are Holtzer Cabot polarized loud-ringing type, 300 ohms resistance, operating on 50- or 60-cycle, 110-volt alternating current. Current is taken from a 1,000-volt power circuit and transformed to 110 volts. The three-wire system is used for the bells of each shaft compartment. Where telephones are also installed, both bell and telephone wires are in one wire-wrapped, lead-covered cable containing several reserve wires for emergency use. No. 18 size wires are used. Fig. 9 shows the waterproof pull switch used in signaling, which has given satisfactory service.

Mining Drainage

The main pump station is located on the 18th level near the San Guillermo shaft. The 10-in. discharge column of Mannesman drawn-steel tubing with bolted forged-steel flanges and round copper gaskets, is carried horizontally on concrete pillars to the San Francisco shaft No. 1 and thence vertically to the tunnel level, a total length of 2,067 ft. and a vertical lift of 1,328 ft. The equipment of the station consists of three 500 gal. per minute, horizontal, duplex, double-acting Prescott high-duty pumps, each driven through one reduction of herringbone steel gearing by a three-phase, 50-cycle, 1,040-volt, 250-hp. General Electric induction motor, operating at 480 r.p.m.

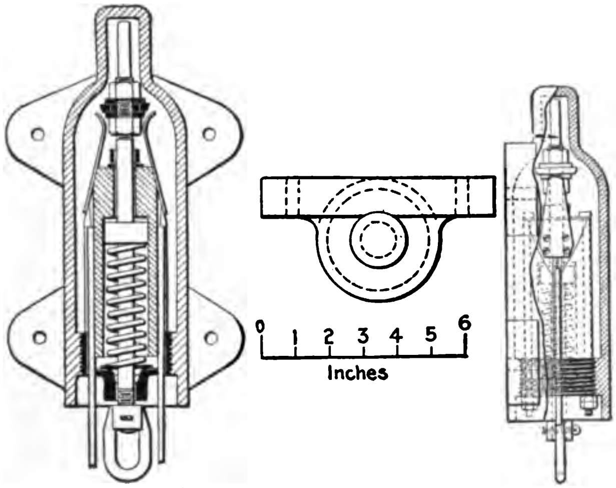


Fig. 9.—Signal Switch.

No water-hammer difficulties have developed during 3 years' service and the equipment in general has given excellent satisfaction. A traveling crane made the installation easy and assists materially in repair work. The pumps were installed under a guarantee of 85 per cent. efficiency based on power delivered to motor pinion. The motor efficiency was measured at the factory. A series of tests made by the local staff, after the pumps had been in operation for over a year, gave the results shown in Table 1.

The old main pump station was at the 15th level and consisted of two eight-stage Sulzer centrifugal pumps each direct connected to 480-

hp. induction motors of German make, no-load speed, 1,200 r.p.m., capacity 3,600 liters per minute each. These motors were designed for 60-cycle current and at 50 cycles failed to handle the water satisfactorily because of reduction in speed. One of these units is now installed on the 18th level and, being connected to the suction of the pump remaining on the 15th level, the two now serve as a reserve unit of about 1,000 gal. per minute capacity.

Water from the 20th level is pumped to the 18th level by a 1,000 gal. per minute Alberger centrifugal pump, electrically driven. In shaft sinking below the 20th level, No. 10 Cameron sinking pumps are used with air at 80 lb. pressure, and as the lift becomes too great for the air pumps, electric centrifugal station relay pumps are installed.

TABLE 1.—*Results of Test on Pumping Equipment*

No. of Pump	Water Pumped, Gallons per Minute	Static Head, Feet	Static Head, Pounds	Dynamic Head, Feet	Dynamic Head, Pounds	Theoretical Water Horsepower	Input to Motor, Horsepower	Motor Efficiency from Curve	Input to Pump, Horsepower	Pump Efficiency, Sump to Discharge	Combined Pump and Motor Efficiency
1	480	1,328	573	1,371	591	160	193	93.1	180	89.2	83.0
2	481	1,328	573	1,371	591	161	199	93.1	186	86.6	80.7
3	469	1,328	573	1,371	591	157	192	93.1	179	88.9	81.6
1 & 2	950	1,328	573	1,388	599	317	408	93.1	380	83.6	77.8
1 & 3	940	1,328	573	1,388	599	314	400	93.1	372	84.4	78.5
2 & 3	939	1,328	573	1,388	599	314	403	93.1	375	83.6	77.8
1, 2 & 3	1,421	1,328	573	1,418	612	475	650	93.1	605	78.4	73.0

Motor data, 480 r.p.m., 250 hp., 1,040 volts.

Pump data, 49.6 r.p.m.; length of stroke, 3 ft.; original diameter of plunger, $4\frac{5}{8}$ in.; diameter of plungers at time of test, $4\frac{9}{16}$ in.; calculated displacement of plungers at $4\frac{5}{8}$ in. diameter, 519.4 gal. per minute; at $4\frac{9}{16}$ in. diameter, 505.5 gal. per minute.

Temperature of water, 87° F.

Weight of water 8.304 lb. per U. S. gallon.

NOTE.—After the tests a small stick was found under one valve in pump 3.

Average gallons per minute pumped, 815; per day, 1,173,600; average pumping cost, including auxiliaries per day, $208.24; per gallon, 0.0177 c.

Ventilation

The natural ventilation of the mine is good, due to the many shaft connections as shown on the longitudinal projection (Fig. 10)—and to communications with eastern adjoining properties, Barron and La Blanca. A motor-driven Sirocco fan is occasionally used temporarily to ventilate some section pending the completion of connections for natural air currents.

Sanitation

Particular attention is paid to keeping all parts of the mine free from refuse or filth. Special sanitary buckets are provided in suitable places on each level which the men are required to use and which are systematically taken to the surface for cleansing, lime being applied liberally underground to limit odors.

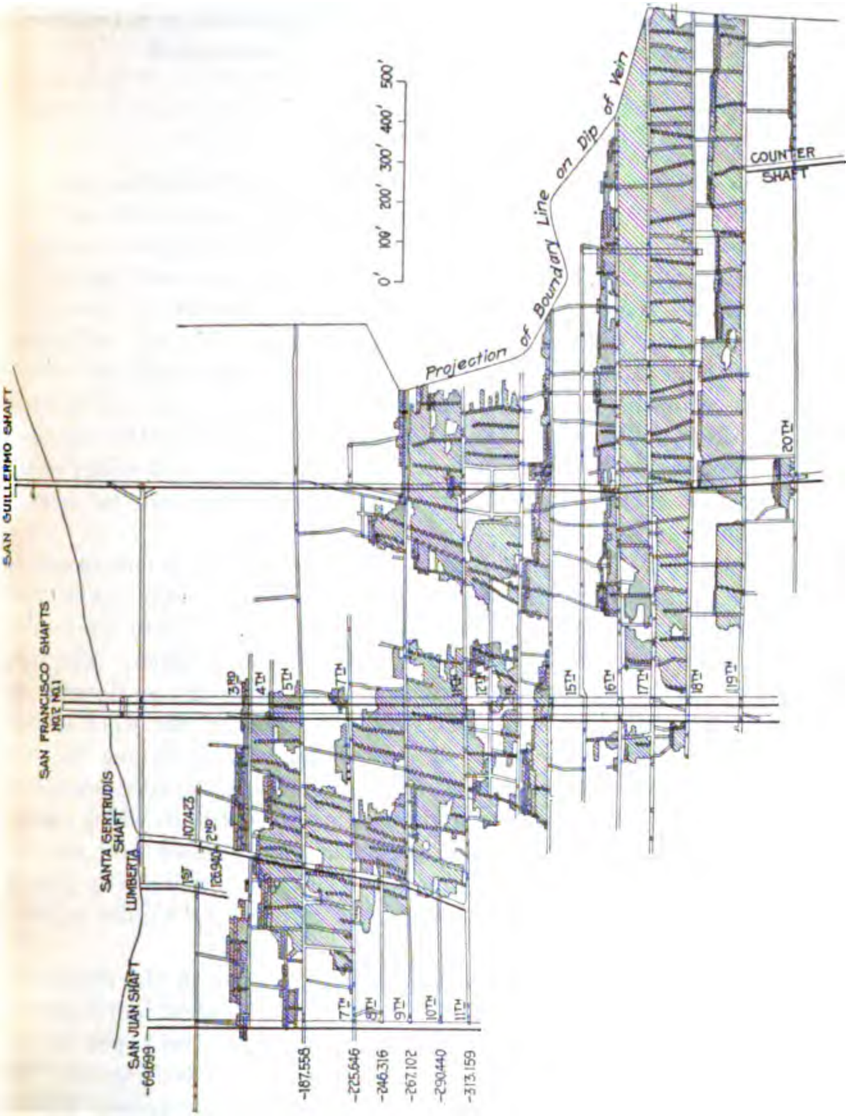


FIG. 10.—VERTICAL PROJECTION. NORTH VEIN.

The main levels, where there is much traffic, are sprinkled with water daily, using a small truck fitted with a barrel having an ordinary sprinkler attachment.

Good drinking water is piped to each shaft station.

A well-ventilated and heated change room is provided, equipped with shower baths of hot and cold water. This facility is not ordinarily found at a mine in Mexico, but its use is to be encouraged.

Safety-First Measures

Within the last two years, an active safety-first campaign has been in hand with very gratifying results. As is well known to all operators in Mexico, the native labor is unusually careless of danger or ordinary precautions against accident. Hence the problem presents exceptional difficulties and requires all the more careful attention to preventive details. Because of the reckless irresponsibility of the individual, coupled with a callousness to accidents which in fairness must be charged to the operators as well as to the men, the accident rate in Mexican mines is abnormally high. And for the reasons stated, this rate will always compare unfavorably with those of well-regulated mines in the United States or Europe. Nevertheless, a great deal can be done to improve this condition.

After securing information from some of the leading American mining companies, experienced in safety-first measures, similar steps were undertaken here. A book of instruction covering accident prevention and first-aid measures was issued in Spanish and English. The shift bosses or "capitanes" of each mine section were made directly responsible for accidents, careful statistics being kept of the work of each boss, with a monthly premium of money to the one having the best record for the month, and a gold watch with suitable engraved inscription for best record for 6 months. An annual contest is held in which various crews compete in rescue and first-aid measures. Bosses and men are drilled in the use of oxygen helmets. A monthly bulletin is printed in Spanish and distributed among the men, in line with similar bulletins issued by American companies.

As of interest to other operators in Mexico, some of the mechanical measures undertaken will be described briefly. All shaft entrances are provided with self-closing gates. After trying out several types, the one selected as most simple and efficient is as illustrated by Fig. 11. The gate is merely swung off center as shown and closes by gravity without undue jar. Fig. 11 also shows the upper gate similarly swung, to protect the individual from falling rocks while standing at the shaft.

Many accidents in Mexican mines are from men carelessly stepping into open chutes or manways in stopes or levels. All chuteways throughout the mine are now covered with rail grizzlies; manways either have a trapdoor of light corrugated iron or, where this interferes with ventilation, the timber is carried up sufficiently so that the top can be covered, leaving an opening at one end or on the footside for passageway. This opening is kept small enough so that a man cannot slip into it inadvertently. All manway openings on levels have, in addition, a warning red electric light.

Explosives to be used currently are stored underground in suitably located crosscuts, fitted with steel doors set in masonry. Fuse and caps are similarly stored separately. It having been found impossible to prevent the practice of crimping the caps with the teeth, fuse is now issued from these underground stores, cut to desired length and with caps properly crimped on by the storekeeper. The main storage of explosives is located some distance from the property.

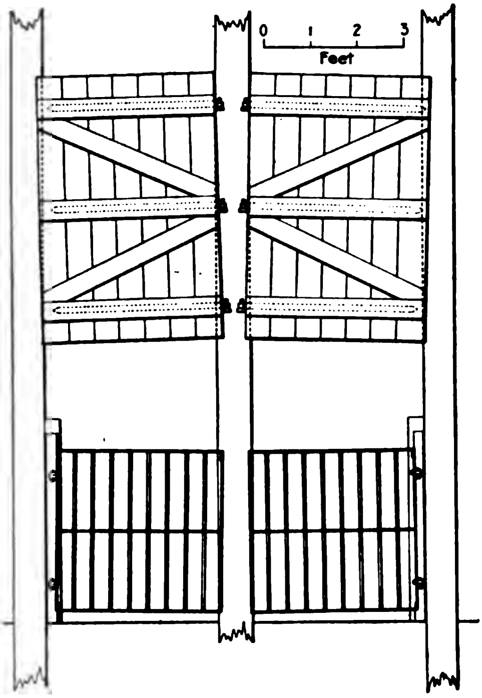


Fig. 11.—Self-Closing Gate Used at Shaft Entrances.

Rock falls are a principal cause of many accidents. As elsewhere, the main preventive is constant vigilance on the part of the bosses combined with such coöperation as it is possible to secure from the men. This spirit of coöperation is obtainable only in a limited amount in Mexico, but continued application will bring it forth to a helpful extent.

Following are accident statistics per 10,000 shifts, before and after safety-first measures were undertaken:

Accidents

	Deaths	Requiring Operation	Trivial	Total
Before	0.574	1.436	12.067	14.077
After	0.334	0.669	8.358	9.361

NOTE.—Trivial accidents include numerous bumps, scratches, slight cuts, etc., which do not incapacitate men from working.

MINING COSTS

For convenience of comparison and of preserving complete records in compact form, the monthly costs are entered upon a tracing form from which blueprint copies are taken. The system requires perhaps less work than where similarly detailed costs are typewritten, whereas the advantage is obvious over any method which does not permit continuous column comparison of items.

Average costs are given below. These do not include recent results which, because of special influences, have been too low to present fairly what should be obtainable under normal conditions:

	Per Dry Short Ton
Development	$0.52
Ore breaking	1.47
Mine to mine bins	0.24
Drainage	0.24
Surface expense	0.04
General expense	0.27
Total mining cost	$2.78

These results are on the basis of a daily production of 1,000 short tons of ore and may be taken as fairly representative of the Pachuca district. In the adjoining Real del Monte district, lower costs are obtainable because the harder walls permit the adoption of the shrinkage system of stoping with consequent reduction, if not elimination, of expenditure for square-set timbering and filling. At Santa Gertrudis, for each ton of ore extracted, two-thirds of a ton of waste must be supplied and handled for filling, and 17 ft. board measure of timber used. The shrinkage system affords another important advantage in Mexico by providing a reserve of broken ore which may be drawn upon more heavily at times to make up for current deficiencies due to numerous feast days and a relatively low constancy of labor. Another important factor affecting local mining costs is the light weight of ore in place and filling ore (old fills). The former averages 13.6 cu. ft. per ton; the latter averages 17.2 cu. ft. per ton. At Santa Gertrudis, the moisture in the ore averages 5.0 per cent., and filling ores represent 65 per cent. of the total tonnage mined.

A fair grade of timber is obtainable at $20 per 1,000 ft. board measure. Other supplies, if carefully purchased in quantities, cost about the same as in a mining district in Colorado or Nevada.

Labor efficiency is low, the complicated mining system likewise adversely affecting the duty per man. Including all labor in stopes for drilling, timbering, filling, etc., 1.27 tons of ore per man-shift are broken and placed in the chutes; based upon all underground labor, including bosses and foreman, 0.687 ton of ore per man-shift is delivered to the surface.

Power is purchased at from $45 to $50 per horsepower-year from the Mexican Light & Power Co. whose hydro-electric plant is about 75 miles away.

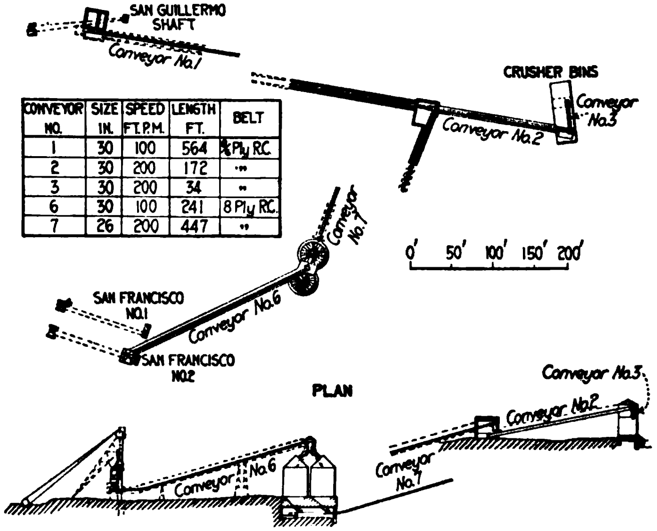
SURFACE TRANSPORTATION

All ore is delivered from the mine by self-dumping skips, discharging into surface ore bins. Thence it is transported to the crushing-plant ore bins by means of belt conveyors. Figs. 12 and 13 show in plan the conveying system from the San Francisco No. 2 and San Guillermo shafts to the crusher bins, and also a cross-section of the line from San Francisco shaft No. 2.

Because of the previously mentioned free-running quality of the ore, plus the unusual percentage of fines which reach the belts first and form a cushion or bed for the larger rocks, the system operates with minimum wear or trouble. Hand-operated sector gates are used throughout, the chute lips being hinged and counterweighted to swing up from the belt when not in use.

On account of re-mining old fills, a large amount of decayed wood comes up with the ore and is picked off at the belts. Careful sorting tests have proved that waste picking cannot be done at a profit. Under close supervision, 3.5 per cent. by weight was picked off, averaging $0.70 per ton at a cost of $0.30 per ton of waste; increasing the waste sorted out to 5 per cent. resulted in an increase of value to $1 at a cost of $0.40 per ton of waste. The actual cost of passing these grades of material through the mill, less value recovered, is calculated to be lower than the sorting cost plus recoverable value lost. Therefore, waste sorting has not been adopted. Sorting high-grade ore for shipment to smelter, because of small quantity available as against high milling efficiency, is not profitable.

This "Mine Bins to Mill Bins" cost has averaged $0.02 per dry short ton. Belt wear is exceedingly low. Belt No. 2 (Fig. 12) is the only one which has been replaced once against a total of 1,131,708 tons handled to date. Belt No. 1, originally installed and put into operation in June, 1911, has handled 691,999 tons of ore and from all appearances is good for a repetition of this duty. Belts 6 and 7, a later installation, have been in operation since May, 1913, handling a total of 439,709 tons and showing little wear.



CONVEYOR NO.	SIZE IN.	SPEED FT.P.M.	LENGTH FT.	BELT
1	30	100	564	[illegible] Ply RC.
2	30	200	172	"
3	30	200	34	"
6	30	100	241	8 Ply RC.
7	26	200	447	"

Fig. 12.—Plan Conveying System.
Fig. 13.—Section View.

Milling

The milling business is conducted by a separate organization on a custom basis. The capacity of the plant is 1,100 tons per day.

Before taking up details for construction and operation, a condensed description of the mill flow scheme (Fig. 14) will be given.

The crushing-plant ore bins of 600 tons capacity feed two No. 6K gyratory crushers, discharging to a double screen 14 ft. long. The oversize from the 4-in. round-hole screen passes by conveyor to the tube-mill rock-storage bin, while the undersize joins the oversize of the 2-in. round-hole screen for crushing in two No. 4K gyratory crushers. The undersize from the 2-in. screen, together with the discharge from the secondary crushers, is delivered to a 22-in. troughed-belt conveyor, equipped with a Merrick weightometer.

The 22-in. conveyor discharges over the first one of three Vezin samplers, a 5 per cent. cut being taken and stored in a sample bin of 15

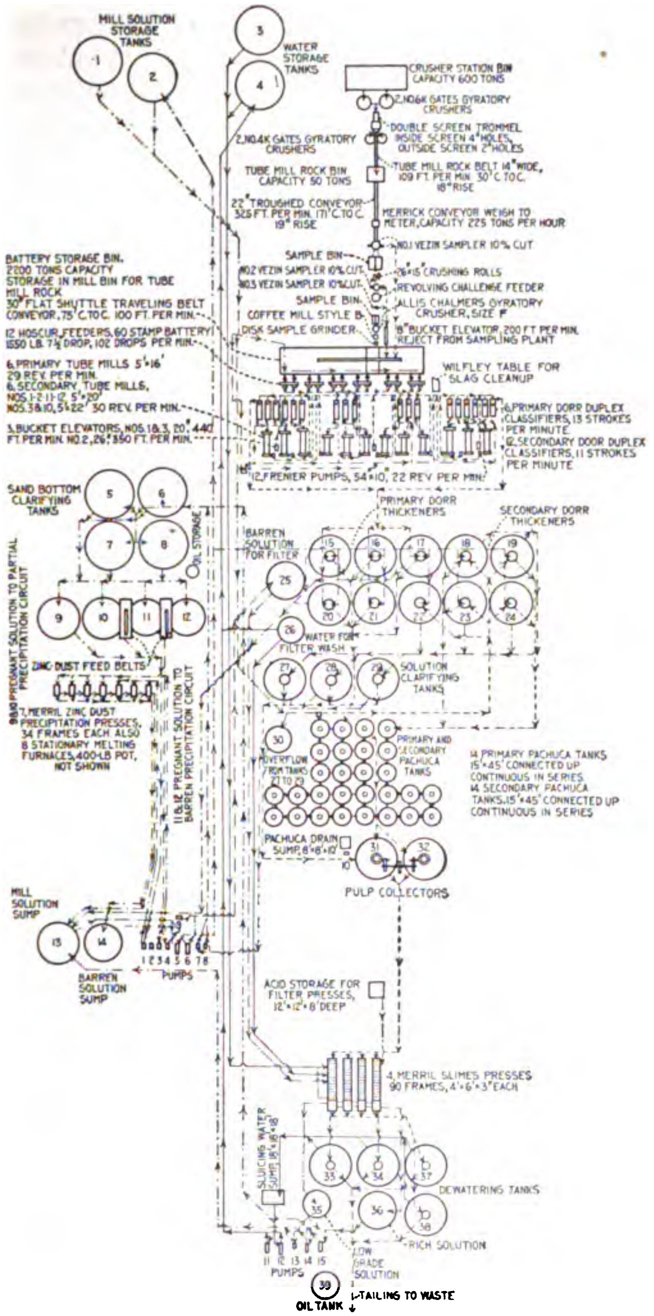

MILL SOLUTION STORAGE TANKS
1
2
3
4
WATER STORAGE TANKS
CRUSHER STATION BIN CAPACITY 600 TONS
2, NO.6K GATES GYRATORY CRUSHERS
DOUBLE SCREEN TROMMEL INSIDE SCREEN 4" HOLES, OUTSIDE SCREEN 2" HOLES
2, NO.4K GATES GYRATORY CRUSHERS
TUBE MILL ROCK BIN CAPACITY 50 TONS
TUBE MILL ROCK BELT 14" WIDE, 109 FT. PER MIN. 30' C. TO C. 18" RISE
22" TROUGHED CONVEYOR 325 FT. PER MIN. 171' C. TO C. 19" RISE
MERRICK CONVEYOR WEIGH TO METER, CAPACITY 225 TONS PER HOUR
NO.1 VEZIN SAMPLER 10% CUT
SAMPLE BIN
NO.2 VEZIN SAMPLER 10% CUT
NO.3 VEZIN SAMPLER 10% CUT
26"×15" CRUSHING ROLLS
REVOLVING CHALLENGE FEEDER
SAMPLE BIN
ALLIS CHALMERS GYRATORY CRUSHER, SIZE F
COFFEE MILL STYLE B
DISK SAMPLE GRINDER
18" BUCKET ELEVATOR, 200 FT PER MIN. REJECT FROM SAMPLING PLANT
BATTERY STORAGE BIN. 2200 TONS CAPACITY STORAGE IN MILL BIN FOR TUBE MILL ROCK
30" FLAT SHUTTLE TRAVELING BELT CONVEYOR, 75' C. TO C. 100 FT. PER MIN.
12 HOSCUR FEEDERS, 60 STAMP BATTERY 1550 LB. 7½" DROP, 102 DROPS PER MIN.
WILFLEY TABLE FOR SLAG CLEANUP
6 PRIMARY TUBE MILLS 5'×16' 29 REV. PER MIN.
6 SECONDARY TUBE MILLS, NOS. 1-2-11-12, 5'×20' NOS. 3 & 10, 5'×22' 30 REV. PER MIN.
6 PRIMARY DORR DUPLEX CLASSIFIERS, 13 STROKES PER MINUTE
12 SECONDARY DORR DUPLEX CLASSIFIERS, 11 STROKES PER MINUTE
3 BUCKET ELEVATORS, NOS. 1 & 3, 20", 440 FT. PER MIN. NO. 2, 26", 350 FT. PER MIN.
16"
12 FRENIER PUMPS, 54"×10", 22 REV. PER MIN.
SAND BOTTOM CLARIFYING TANKS
5
6
7
8
OIL STORAGE
PRIMARY DORR THICKENERS
SECONDARY DORR THICKENERS
BARREN SOLUTION FOR FILTER
15
16
17
18
19
20
21
22
23
24
25
26
WATER FOR FILTER WASH
9
10
11
12
9 & 10 PREGNANT SOLUTION TO PARTIAL PRECIPITATION CIRCUIT
ZINC DUST FEED BELTS
7 MERRIL ZINC DUST PRECIPITATION PRESSES, 34 FRAMES EACH ALSO 8 STATIONARY MELTING FURNACES, 400-LB. POT, NOT SHOWN
11 & 12 PREGNANT SOLUTION TO BARREN PRECIPITATION CIRCUIT
27
28
29
SOLUTION CLARIFYING TANKS
30
OVERFLOW FROM TANKS 27 TO 29
PRIMARY AND SECONDARY PACHUCA TANKS
14 PRIMARY PACHUCA TANKS 15'×45' CONNECTED UP CONTINUOUS IN SERIES
14 SECONDARY PACHUCA TANKS, 15'×45' CONNECTED UP CONTINUOUS IN SERIES
PACHUCA DRAIN SUMP, 8'×6'×10'
10
31
32
PULP COLLECTORS
MILL SOLUTION SUMP
13
14
1 2 3 4 5 6 7 8
PUMPS
BARREN SOLUTION SUMP
ACID STORAGE FOR FILTER PRESSES, 12'×12'×8' DEEP
4 MERRIL SLIMES PRESSES 90 FRAMES, 4'×6'×3" EACH
SLUICING WATER SUMP 18'×18'×18'
33
34
37
DEWATERING TANKS
35
36
38
RICH SOLUTION
11 12 13 14 15
PUMPS
LOW GRADE SOLUTION
39
OIL TANK
TAILING TO WASTE

FIG. 14.—MILL FLOW SHEET.

tons capacity. The reject flows to a 30-in. shuttle-type flat-belt conveyor for distribution into battery bins of 2,000 tons capacity. The first sample cut is fed from the 15-ton bin through a set of 26 by 15-in. rolls reducing to $\frac{3}{4}$ in. and finer, and discharging to a second Vezin sampler cutting out 10 per cent. which is delivered by a revolving Challenge feeder to the third Vezin taking a 20 per cent. cut. The sample thus obtained is 1 ton for each 1,000 tons milled and is crushed to $\frac{1}{4}$ in. in a size F gyratory crusher, thereafter being cut down by Jones riffles and reduced in the usual manner. Rejects from the second and third Vezin samplers and from the quartering floor are returned to the battery bins by an 8-in. elevator.

The ore passes from the battery bins through Hoscur feeders to 60 1,550-lb. stamps arranged in units of 10 stamps each, 20 stamps being driven by a 65-hp. motor, belted to a jack shaft; the stamps make 102 $7\frac{1}{2}$-in. drops per minute; 3-mesh and 4-mesh screens are used, the pulp from the batteries flowing through split distributing launders to six primary duplex Dorr classifiers, the sand passing to six 5 by 16-ft. tube mills, the discharges from which are delivered to the launder leading to the secondary classifiers by elevator or by reserve 10 by 54-in. Frenier pumps, one pump to each mill. The slime overflow from the primary classifiers is laundered to eight secondary duplex Dorr classifiers, which feed the sand to four 5 by 20-ft. and two 5 by 22-ft. tube mills, the discharges from which are returned by elevator or Frenier pumps to the classifiers.

The slime overflow from the secondary classifiers passes to six primary Dorr thickening tanks, 35 ft. diameter by 15 ft. deep, wherein the pulp is thickened from 10 to 1 to 1.5 to 1.

The thickened pulp is delivered to a set of 14 primary Pachuca tanks, 15 by 45 ft., operating in series; the discharge from the last tank in the series is by means of an air lift, submerged in the tank itself, and delivering to a launder where a wash of four parts of mill solution is applied, the pulp flowing to four secondary Dorr thickening tanks, 35 by 15 ft., wherein it is again thickened to about 1.75 to 1 and delivered to a set of 14 secondary Pachuca tanks, 15 by 45 ft., also operating in series, the discharge from the last tank of the series flowing to two 35 by 12-ft. storage tanks equipped with mechanical agitators.

These storage tanks feed by gravity at 35 lb. pressure four 90-frame Merrill filter presses, size of leaf 4 ft. by 6 ft., width of frame 3 in. The tailing sluiced from these presses flows to four 35 by 15-ft. Dorr thickening tanks for recovery of water before passing to the tailing storage dams.

The pregnant solution is clarified by passing through four sand filter tanks, 40 ft. diameter by 10 ft. deep.

Precipitation is effected by the Merrill zinc-dust process in two circuits, partial and barren, in order to economize in zinc dust. Two presses are in use on the barren circuit and three on the partial.

The precipitate is melted in a battery of eight oil-fired No. 400 crucible furnaces arranged on the arc of a circle and served by a radial jib crane, fitted with an air cylinder for raising and lowering the pots.

All pumping of solution and water is concentrated in two pump houses, one located below the filter plant and the other below the precipitation plant. The pumps are of vertical triplex plunger types, gear-driven by motors, and are installed so that one pump is in reserve for two circuits.

An electrically operated inclined tramway runs from the top to the bottom of the mill delivering material to any floor.

DETAILS OF CONSTRUCTION AND OPERATION

Compactness of design was sacrificed in order to secure a gravity flow. Such design was permissible owing to an unusually ample mill site of 17° slope, coupled with a mild climate requiring no housing of tanks. Supervision is made easy because the size of the plants warrants the division into two departments, mill and cyanide, and the tramway is used by the bosses in getting round.

The mill is electrically driven throughout, 50-cycle alternating current being distributed at 440 volts.

Crushing, Weighing and Sampling

The crushing-plant design exhibits no points of special interest. The elimination of elevator returns and the favorable character of the ore make the operation unusually easy and simple. Consumption of steel liners, etc., is almost negligible. The 6K gyratory crushers, driven by 30-hp. motors, take only 15 hp. each. This is again due to the character of the ore and the large percentage of fines which, if the ore had greater abrasive quality, it might pay to screen out beforehand.

An ample supply of mine rock, over 4 in. in size, for use in the tube mills in place of pebbles, is obtained cheaply from the revolving-screen oversize as described above.

The Merrick weightometer is checked weekly against a weighed quantity of ore, the average error being well under 1 per cent. Current weighings are corrected by the weekly factor thus obtained.

Particular attention is paid to securing an accurate sample of the ore delivered to the mill. Ordinarily two classes of ore are sampled separately each day. The rejects from these two are thoroughly mixed and quartered down, to make the mixture sample for the day. The calculated average assay of the two class samples checks very closely the assay results of the mixture. The usual samples are taken for moisture, which averages about 5.0 per cent.

Stamping and Tube Milling

The design of the stamp battery follows standard lines except in a few minor details. The mortar-box foundation bolts are crossed as shown in Fig. 15, thereby permitting a broken bolt to be removed easily; none has broken thus far. A traveling crane, as well as a crawl, installed with the building, was found very useful in erecting the battery and in making subsequent current repairs.

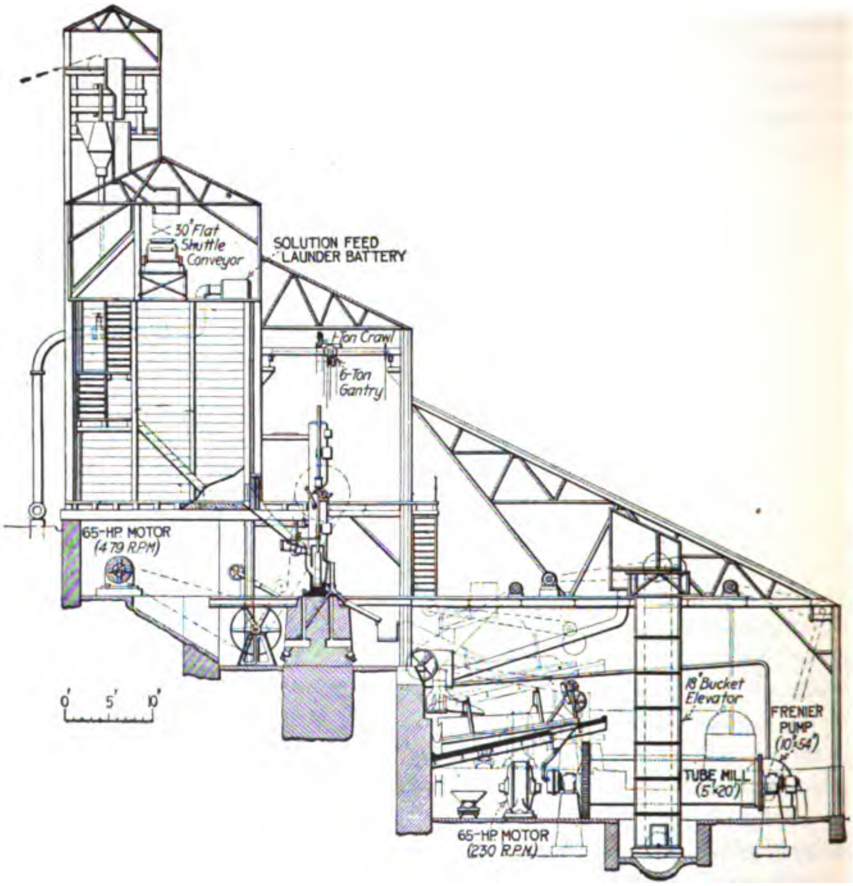


FIG. 15.—STAMP MILL. SECTIONAL VIEW.

The stamp duty averages 21.1 tons through 3-mesh and 4-mesh screens. Crushing is in mill solution, 10 parts of solution to 1 of ore. Screen wear is of no importance. Steel wear per ton is: Shoes 0.16

lb., dies 0.08 lb., liners 0.60 lb. Life of shoes, 9¼ in. diameter by 14 in. long, averages 90 days; of dies 97 days; of liners 43 days. Shoes and dies are forged steel. Liners are cast iron made locally. Manganese-steel liners have been tried, but the cost per ton crushed was considerably higher. The 65-hp. motors, each driving 20 stamps, are overloaded about 8 per cent.

A trial was made of introducing the battery solution through nozzles in the back of the mortar box above the dies, but without success. Stationary screens placed between the feeder and the mortar box to take out fines were also tried and abandoned, the benefit being doubtful while requiring more supervision.

Average screen analyses of feed and discharge of the stamp battery, equipped with 3-mesh, No. 32 wire screens, are:

	Feed, Per Cent.	Discharge, Per Cent.
Inches		
+ 2	0.5	
+ 1	18.4	
+ ¾	13.8	
+ ½	8.0	
Mesh		
+ 4	18.7	4.0
+ 8	11.5	16.3
+ 10	1.8	7.5
+ 20	8.5	14.8
+ 30	3.9	8.1
+ 40	1.4	4.4
+ 60	2.2	6.7
+ 80	2.0	6.2
+100	0.5	2.7
+120	0.8	3.2
+150	0.4	1.7
+200	0.2	2.7
−200	6.9	21.2
	99.5	99.5

Distribution of the pulp from the battery to the primary classifiers by means of split launders could be improved by a mechanical distributor of one of the successful revolving types.

The classifier platforms were built on an incline to save headroom over the tube-mill gears as well as to follow the slant of the classifiers. Returns to the classifiers from the tube mills are by elevators with individual Frenier pumps as reserves. A better design following later practice would be to effect such returns by means of the classifiers themselves.

Tube-mill grinding is done in two stages, using 5-ft.-diameter mills throughout, the primary series being 16 ft. long and the secondary series 20 ft. and 22 ft. Comparison of this system with single-stage grinding has failed to show conclusive results in its favor, although a slight benefit is apparent. This benefit is, however, insufficient to warrant a repetition of this refinement of design unless in conjunction with water concentration not required with this ore.

Danish flint pebbles were used for a considerable period, but their increasing cost led to the adoption of mine rock entirely to replace the pebbles. The mine rock supply is obtained mechanically in the crushing plant as described and is sent separately over the regular conveyors to a compartment in the battery bin from which it is transferred by chute to the primary tube-mill floor where it is distributed by car. Part of the rock is introduced into the mill through the feeder. As the trunnion opening of the mills is not as large as it should be, rocks over 5 in. in size as well as occasional large boulders, 12 to 15 in., required in the primary mills for efficient grinding, are loaded into the mills through the manholes once a day; 130 lb. of mine rock are required for each ton of ore milled and are credited to the total tonnage treated.

Tests are now in hand using cast-iron balls in place of mine rock. Results thus far obtained indicate a capacity increase of 33 per cent. with finer grinding. Power load shows an increase of 33 per cent., from 65 to 90 hp. per mill. Forged-steel balls ordinarily used for such grinding were not obtainable but it is quite probable that chilled cast-iron or semi-steel balls and liners will prove more economical, taking into account the low cost of locally made castings, 2.5 c. per pound, as against high first cost plus importation expenses of steel balls. Ball wear is 1.7 lb. per ton milled. Tube-mill liners are modified El Oro type, of hard cast iron, the average life being 6 months; cost 2.2 c. per ton of ore milled.

The motor driving each mill through a flexible coupling and one reduction of spur gearing is of high torque induction type, 65-hp., 230 r.p.m. Allis Chalmers make. Its ample design easily carries the overload. But the power factor of this motor is low, 75 per cent., and an improvement would be to use a higher-speed motor with a single reduction of herringbone gearing.

The economical grinding point is taken at 75 per cent. through 200-mesh. Table 2 gives screen tests of feed and discharge of primary and secondary tube mills, using mine rock and at a plant capacity of 1,000 tons per day.

Agitating

An extraction of 55.5 per cent. of the gold and 18.6 per cent. of the silver takes place in the mill before the pulp reaches the Pachuca tank system.

TABLE 2.—*Screen Tests of Tube-Mill Feed and Discharge*

Mesh	Primary Feed, Per Cent.*	Tube Mill Discharge, Per Cent.	Secondary Feed, Per Cent.†	Tube Mill Discharge, Per Cent.
+ 4	13.9	0.2	0.8	1.6
+ 8	22.8	4.5	0.5	0.1
+ 10	8.6	2.2	0.4	0.2
+ 20	20.6	8.5	1.6	0.2
+ 30	11.8	10.9	3.4	0.7
+ 40	4.7	4.8	3.3	1.0
+ 60	8.0	12.1	17.5	10.8
+ 80	1.9	10.2	9.6	5.6
+100	1.6	5.2	11.4	10.3
+120	1.5	6.5	17.6	15.5
+150	0.7	4.0	9.5	11.8
+200	0.8	5.7	8.4	11.2
−200	2.5	25.0	15.6	30.7
	99.4	99.8	99.6	99.7

* Moisture, 35 to 40 per cent. Operating without return, 175 tons of ore pass through tube mill per 24 hr.

† Moisture, 35 to 40 per cent. Operating with return, in closed circuit, 200 tons of ore pass through mill per 24 hr.

Secondary Classifier Overflow

(Finished Product of Mill)

Mesh	Per Cent.
+100	3.8
+120	5.7
+150	5.5
+200	9.5
−200	75.1
	99.6

Cyanide of sodium, either 128 or 120 per cent. as obtainable, in lump or in brick form, is added at the first tank of the primary series, at the rate of 4 g. sodium cyanide for each gram of silver in the ore delivered to the mill. The sodium cyanide consumed is 3.15 lb. per ton, including mechanical loss. The strength of solution at the beginning of agitation is 0.55 per cent. KCN; of the mill solution, 0.4 per cent.

Protective alkalinity is maintained at about 0.75 per cent.; the lime, fed dry into the ore at the crushing plant, is low-grade, averaging about 65 per cent. available, the consumption being 20 lb. per ton. Arrangements are in hand to improve the method of feeding by emulsifying the lime, adding it either to the primary tube mills or to the Pachuca tanks.

Crude litharge, between 85 and 90 per cent. PbO, adopted in place of lead acetate as both cheaper and more efficient, is ground in a small tube mill, 24 in. diameter by 37 in. long, discharging into the first Pachuca tank at the rate of 0.6 lb. per ton of ore.

The best results are obtained with 72 hr. agitation although 60 hr. give within 2 per cent. as high extraction. The air pressure is 27 lb., 75 cu. ft. per minute being required for each tank.

No difficulty has been experienced with the series operation of the Pachuca tanks. Connections between tanks are made by 10-in. horizontal pipes, located 3 ft. from the top, the joint between two abutting pipes being made leak proof by a wrapping of tarred canvas. This joint takes up the tank vibration and effects an easy connection.

To avoid undue accumulation of slime on the inside of the tanks, they are emptied and sluiced down once a month. This operation requires about 3 hr. per tank, using a 3-in. Traylor slime pump for the return.

Screen tests of the inflow and outflow of the system are practically identical, showing that there is no segregation or short-circuiting of sand or slime.

Filtering

This step in the process is fairly difficult, due to fine grinding of ore containing a considerable amount of colloidal matter. Also, the dissolved values in the pulp to the filters are high, 3.5 oz. silver per ton of solution. After extended working-scale tests of several types of vacuum and pressure filters, Merrill presses were adopted, using the center system of filling.

A press cycle occupies 75 min. made up of:

	Min.
Charging	26
Barren solution wash	13
Water wash	13
Sluicing	23
	75

Bristol recording pressure gages are attached to the filling pipe of each press, the cycle curves from the chart giving an excellent check upon the care taken by the attendants in operating the presses. The sluice valve bar and the filling valves are electrically connected so that both cannot be coincidently opened without ringing an alarm bell. The cakes average 1¼ in. in thickness, thereby leaving ½ in. space in the center of the 3-in. frame for entrance of washes. Dry pulp handled per cycle, 14.5 tons.

Sluicing water is at 90 lb. pressure and five parts are required to clean out a press. The bulk of this water is recovered in dewatering tanks as explained heretofore.

No. 6 cotton duck, 72 in. wide, is used regularly, after extensive trial of several other weights. A set of cloths lasts about 2,500 cycles. Acid washing to remove lime is done in the press every 10 days, using a 0.75 per cent. sulphuric acid solution.

Much care is taken with the nozzles of the sluice bar. The bar of each press is taken out and tested every 48 hr., any defective nozzles being replaced. Nozzles are ordinary ½-in. cast-iron plugs, drilled $\frac{3}{16}$-in. hole; these last practically as long as special steel nozzles and are very much cheaper.

The unwashed values in the press discharge average a trace of gold and 0.08 oz. silver, showing an efficiency of 97.7 per cent.

Clarifying Solutions

All solutions to be precipitated are first passed through ordinary sand filter tanks. The addition of about 40 per cent. by volume of sawdust to the sand considerably improves the clarifying efficiency of the filtering medium and at the same time reduces the frequency of slime skimming and sand cleaning. The latter operation is effected by shoveling the sand from the tank to a launder leading to a small trommel where the sand is washed free of slime, with small loss, and returned for re-use.

In this way, 4,500 tons of solution are filtered daily at a cost of 0.28 c. per ton of solution. Possibly Merrill clarifying presses would be an improvement, although no comparative figures are available. At any rate, the existing equipment does the work efficiently and cheaply, with little supervision required.

Precipitating

The Merrill zinc-dust process of precipitation has in every way justified its adoption. Its efficiency, safety, cleanliness, etc., compared to the zinc-shaving method are too widely known to need further elucidation here. The ease and rapidity with which a cleanup can be made gives it a further special advantage in a good-sized mill treating silver ores with the consequent considerable production of precipitate. Formerly the process enjoyed the further advantage of a lower price for zinc dust than for zinc shavings; the difference now is practically negligible.

The solution from the sand filters averages 0.015 oz. gold and 3.0 oz. silver. Two circuits are maintained, one precipitating to barren solution, sufficient for the filter press washing, and the other circuit to an effluent containing from 0.9 to 1.2 oz. silver per ton. To accomplish this a slight excess of zinc dust is fed to the barren circuit in the proportion of 1.1. oz. zinc dust to 1.0 oz. fine bullion; to the partial circuit 0.8 to 1.0. The average zinc consumption of the plant is 1.0 oz to 1.0 of fine bullion.

Cleanups made at the middle and end of month and sometimes oftener, provide an accurate check on current work.

Melting

The only drying given the precipitates is by blowing air through the presses at 25 lb. pressure for 1 hr., reducing the contained moisture to about 30 per cent. The precipitate is cleaned from the presses into rectangular, steel cars, 5 ft. 10 in. by 6 ft. 4 in. by 1 ft. deep, in which it is weighed. Based upon the calculated dry weight of the precipitate, flux consisting of 1½ per cent. each of borax and bottle glass, is added on top without mixing, and the charge is then shoveled from the car into No. 400 crucibles. To permit the introduction of a high column of precipitate, a discarded pot with the bottom out is temporarily fitted into the crucible, being removed after the charge has melted down. This procedure reduces dust losses and accelerates charging and melting.

The eight furnaces (Fig. 16) are oil-fired, built flush with the floor of the melting room, on the arc of a circle with fire pit at rear. This arrangement has proved very satisfactory and is to be recommended as convenient and labor saving.

The crucible, upon being lifted from the furnace by tongs attached to ½-in. wire cable raised or lowered by air cylinder on the post of the jib crane, is swung around to one side to the pouring carriage containing the bullion molds. The slag is first skimmed off, using sand to assist in this operation. As the bars are poured, sticks of wood are laid on top and, igniting, prevent too rapid cooling of the center of the bar with the consequent subsidence and holing.

The precipitate averages 85 per cent. fine bullion; the Doré bars weigh 1,000 oz. and assay 5.0 gold and 940 silver fine. The No. 400 crucibles contain an average of seven bars per melt with a maximum of 10 bars. The average life of a crucible is 10 melts or 70 bars.

With bullion of this fineness, no difficulty with matte formation is experienced. The bars are carefully cleaned and all chips and corners removed before sampling and weighing. Drill samples are taken, one on the top of the bar, one-third the distance along the diagonal toward the center, and one on the bottom of the bar at one corner; each hole is drilled halfway through the bar.

The bullion was formerly refined by a French concern in Mexico City, the fine gold being sold to the Mexican Government and the fine silver shipped for sale in New York or London. The refinery has been closed down since the outbreak of the war in Europe, so that the Doré bars are now refined and sold in New York or London.

Melting the precipitate in an electric furnace, built after the Alaska-Treadwell and Lluvia de Oro design, is under trial. Alternating current at 120 volts is used, the input being about 600 amp. working at full heat. Trouble has been experienced with dusting and with burning a hole through the bottom of the furnace; a renewable iron plug (Fig. 17) may

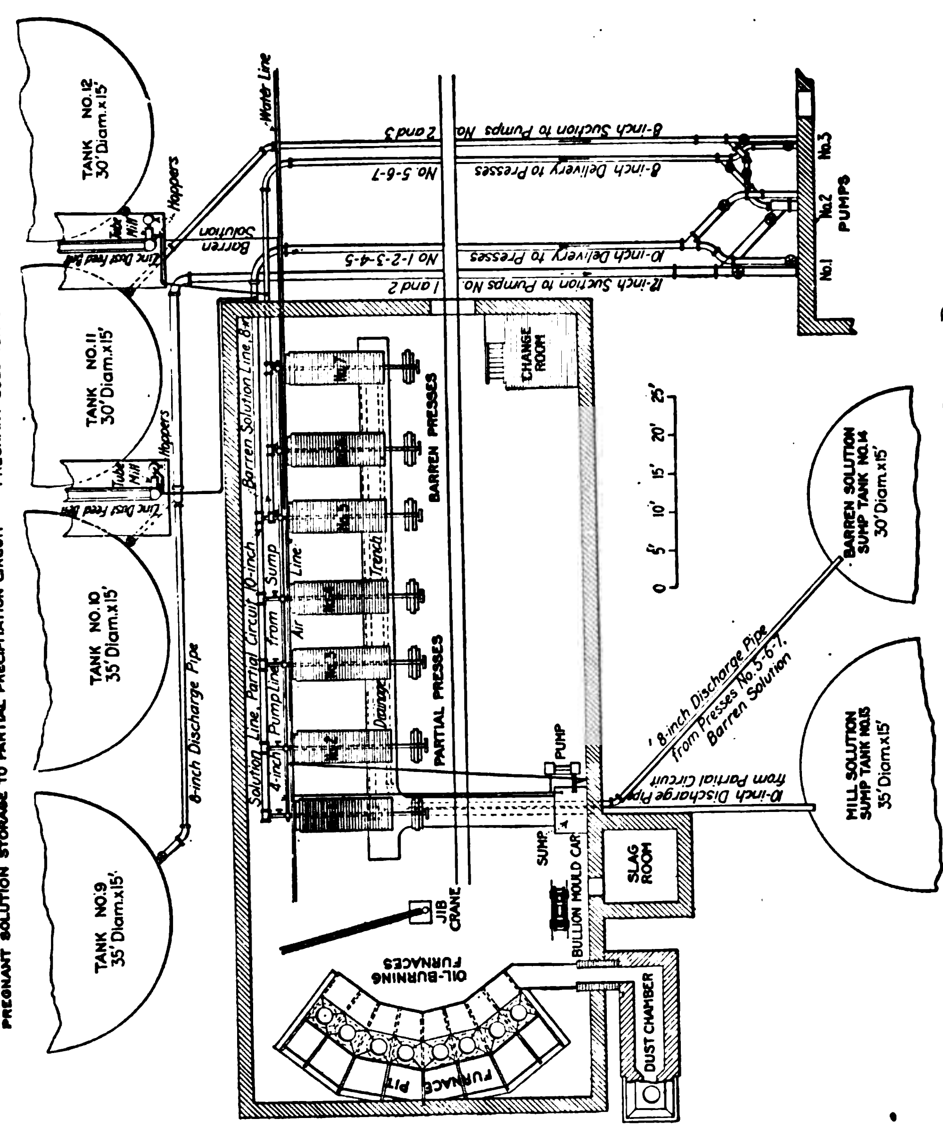

PREGNANT SOLUTION STORAGE TO PARTIAL PRECIPITATION CIRCUIT
PREGNANT SOLUTION STORAGE TO BARREN PRECIPITATION CIRCUIT
TANK NO.12 30' Diam.x15'
TANK NO.11 30' Diam.x15'
TANK NO.10 35' Diam.x15'
TANK NO.9 35' Diam.x15'
Tube Mill
Zinc Dust Feed Belt
Hoppers
8-inch Discharge Pipe
Water Line
Barren Solution
8-inch Suction to Pumps No. 2 and 3
8-inch Delivery to Presses No. 5-6-7
10-inch Delivery to Presses No. 1-2-3-4-5
12-inch Suction to Pumps No. 1 and 2
No.1
No.2
No.3
PUMPS
CHANGE ROOM
Barren Solution Line, 8-inch
Solution Line, Partial Circuit 10-inch
4-inch Pump Line from Sump
Air Line
Drainage Trench
BARREN PRESSES
PARTIAL PRESSES
PUMP
SUMP
BULLION MOULD CAR
JIB CRANE
OIL-BURNING FURNACES
FURNACE PIT
DUST CHAMBER
SLAG ROOM
0 5' 10' 15' 20' 25'
BARREN SOLUTION SUMP TANK NO.14 30' Diam.x15'
MILL SOLUTION SUMP TANK NO.13 35' Diam.x15'
8-inch Discharge Pipe from Presses No. 5-6-7. Barren Solution
10-inch Discharge Pipe from Partial Circuit

FIG. 16.—PLAN OF PRECIPITATING AND MELTING ROOM.

provide a convenient remedy for the latter. The dusting is probably due to the precipitate being wet, the intense heat at the electrode setting up a center draft of steam and air. Either the precipitate should be dried or some dust-collecting apparatus attached to the furnace top. The advantages of this system over the oil-fired crucible method of melting are rapidity of melting and discontinuance of the use of crucibles and oil, both of which require a long distance haul; no data are available yet as to the comparative economy.

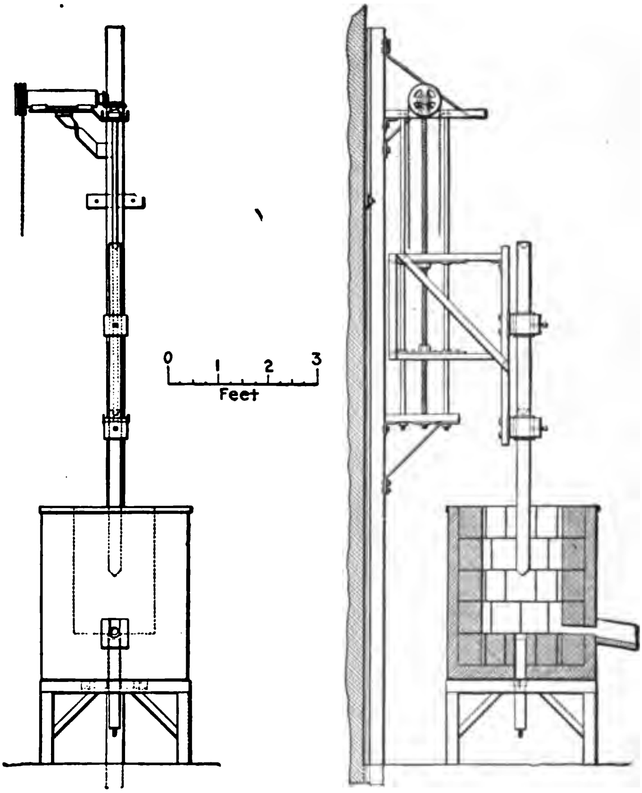


FIG. 17.—ELECTRIC MELTING FURNACE.

Tailing Disposal

As previously described, the tailing sluiced from the Merrill filter presses passes through four Dorr thickening tanks for recovery of water and is then automatically sampled before flowing to the storage basins. These basins or ponds, four in number, are located on the nearest flat land 2 miles away. The ditch thereto was dug U-shaped, 2½ ft. deep with a 3 per cent. grade; 1½ per cent. grade would suffice.

The walls of the storage basins are built up as required of tailing mud, one basin receiving the entire inflow while the others are being prepared. Evaporation and particularly absorption of the solution are high, the decanted effluent representing only 30 per cent. of the solution inflow.

This solution contains an average of a trace of gold and 0.12 oz. silver, representing filter losses and further dissolved values. A small precipitation plant is being arranged to recover this escaping fraction.

Construction Costs

On the basis of 1,100 tons daily capacity the total cost of the plant was $792 per ton capacity. This covered first-class construction methods throughout, designed for long life with minimum maintenance and renewals. A comparatively excessive amount of equipment was required due to fine grinding and especially to length of treatment necessary for a silver ore, resulting in unusually large tankage. On the other hand, the all-sliming design without concentration simplified operations and eliminated the installation of concentration equipment.

EXPERIMENTAL WORK

Aluminum Dust Precipitation[3]

Upon the publication of results obtained at Nipissing with desulphurization and aluminum-dust precipitation, similar experiments were undertaken. Desulphurization, as was to be expected, gave no beneficial results on Santa Gertrudis ore, which contains no tellurides, arsenides, etc. On the other hand, aluminum-dust precipitation showed a considerable improvement over the zinc-dust method and its adoption has only been delayed by war conditions practically having put aluminum dust off the market. Comparison of the two methods was based upon normal prices of aluminum and zinc dust of 30 c. and 7 c. per pound, respectively.

As practically all the existing equipment can be used very conveniently in the aluminum-dust process, the problem as worked out depends for commercial success upon an economical method of converting the lime in solution, added during milling for neutralizing and settling, into calcium carbonate with the formation of the necessary caustic soda as a product of this reaction, all prior to precipitation by aluminum-dust. Any lime present in solution during precipitation will be converted to calcium aluminate, lowering the grade of the precipitate and causing difficulty in melting. The conversion and elimination of lime can be accomplished by the addition of carbonate or bicarbonate of soda, as follows:[4]

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[3] Tests conducted at mill by A. W. Hahn.

$$Ca(OH)_2 + Na_2CO_3 = CaCO_3 + 2NaOH$$
$$Ca(OH)_2 + NaHCO_3 = CaCO_3 + NaOH + H_2O$$

In place of commercial carbonate or bicarbonate of soda, laboratory tests showed the probable successful use of an impure local product known as "Tequesquite," obtained from the dried-up bed of Lake Texcoco and containing approximately 56 per cent. carbonate and bicarbonate of soda in about equal quantities. In practice, the "Tequesquite" solution would be added to the pulp flowing to the collecting tanks above the Merrill filter presses.

In experimental work, the aluminum-dust precipitate yields a bullion 975 fine. The consumption of aluminum dust was about one-third that of zinc dust, or 0.34 oz. to 1.0 oz. fine bullion.

The slightly increased cost per ton of ore of aluminum dust is more than offset by decreased chemical consumption of cyanide, the net saving being calculated at $0.15 per ton of ore.

Electrolytic Regeneration of Cyanide[5]

Prior to the favorable conclusions reached in the aluminum-dust tests, a lengthy investigation was conducted in the regeneration of cyanide in the mill solution by electrolysis.

Anything like a comprehensive description of this interesting work cannot be taken up here and only the important features will be outlined.

The principal difficulty presented was in finding an anode of high conductivity that would not dissolve or disintegrate. Graphite, lead, plumbago, carbon, charcoal, magnetite, peroxidized lead, were tried without much success. An alloy of lead with 6 to 9 per cent. antimony was finally selected as fairly satisfactorily filling the requirements. The following conclusions were then worked out:

1. The solution to be treated by electrolysis must be barren of gold and silver to avoid their precipitation. The question of enlarging the

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[4] For chemistry and details of aluminum-dust process refer to:

E. M. Hamilton: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, 95, No. 19, p. 935 (May 10, 1913).

E. M. Hamilton: *Engineering and Mining Journal*, vol. 99, No. 13, p. 568 (March 27, 1915).

S. F. Kirkpatrick: Aluminum Precipitation at Deloro, Ontario, *Engineering and Mining Journal*, vol. 95, No. 26, p. 1277 (June 28, 1913).

James Johnston: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans*, vol. 48, p. 3 (1914).

G. H. Clevenger: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans.*, vol. 49, p. 156 (1914).

[5] Tests conducted at mill by A. W. Hahn.

scope of the process to include electrolytic precipitation of gold and silver was also investigated separately, as hereinafter discussed.

2. The regeneration of cyanide takes place at the cathode, with a destruction of cyanide at the anode which may be principally overcome by maintaining a high protective alkalinity at the anode.

3. A porous diaphragm such as canvas separating the anode and cathode assists in maintaining an alkaline anolyte, circulation of anolyte and catholyte being accomplished individually.

4. Power consumption is affected by resistance of electrodes, electrolyte and diaphragm, by temperature and circulation rate of electrolyte, and by spacing of electrodes.

5. The voltage drop per cell, at a current density of 15 amp. per square foot of anode surface, treating barren solution containing 0.5 per cent. total KCN and 0.115 per cent. CaO, with canvas diaphragm and electrodes spaced 1½ in. apart, is 6 volts at 70° F. Under these conditions the solution flow should be at least 0.25 gal. per square foot of anode surface.

6. The deposit on the cathode analyzed 71 per cent. zinc, with impurities of copper, iron, arsenic and organic matter. The recovery of this zinc might be turned to some commercial advantage.

7. The reaction representing the regeneration of cyanide is presumably as follows:

$$Na_2Zn(CN)_4 + Ca(OH)_2 + \text{Current} = 2NaCN + Ca(CN)_2 + Zn + H_2O + O.$$

8. The net reduction of cost was estimated at $0.12 per ton of ore. The equipment outlay, however, proved to be several times greater than that required to convert the precipitation plant to the use of aluminum dust and as the latter process at the same time indicated a somewhat larger cost reduction without further complexities of plant, its adoption in place of regeneration was decided upon.

Electrolytic Precipitation and Regeneration

Following the experience gained in the experiments with regeneration above described, the next step was to endeavor to combine with it the electrolytic precipitation of gold and silver from solution.

Difficulty was encountered in obtaining a coherent cathode deposit. It was found uneconomical to attempt a complete precipitation, the limit of commercial work being about 1.3 oz. silver remaining in solution. Plant cost would be high. Altogether the proposition did not present sufficiently attractive possibilities to warrant following it up very thoroughly. With the perfection of the diaphragm cell it is probable that both precipitation and regeneration electrolytically could be worked out to a fair success.

Electrolytic Refining of Precipitate

A preliminary investigation has been made of electrolytic refining of precipitate from the present zinc-dust installation. The chief difficulties to be overcome are resistance of the precipitate to the current, purification of the electrolyte, high acid consumption due to impurities, and refining and melting of the gold slime.

Flotation Concentration

As previously stated, the ore is clean with only small quantities of base metals. A representative analysis is: Au, 0.067 oz.; Ag, 12.0 oz.; Pb, 0.10 per cent.; Cu, 0.20; Zn, 0.15; Fe, 4.40; SiO$_2$, 86.67; S, 1.0 per cent.

Most of the silver is present as argentite, Ag$_2$S—0.05 per cent. It is evident, therefore, that little benefit in recovery is to be expected by gravity concentration. At the time of building the plant complete experiments were conducted on a working scale, the recovery by concentration representing from 8 to 10 per cent. of the value of the ore. This did not justify the installation of gravity concentrating equipment plus added cost of operation as against direct cyaniding.

The growing success of flotation concentration in the United States led naturally to local investigation of the problem, which though still in a laboratory stage, already shows encouraging results. If the process is worked out to a commercial success, a detailed description of results should be of interest. Meanwhile a summary of preliminary conclusions is given:

1. A large variety of vegetal and mineral oils have been tested, the best results thus far having been obtained with one-third pine oil or wood creosote and two-thirds Mexican gas oil, such as is used locally in the oil-fired melting furnaces. The pine oils and wood creosote tried are imported, but doubtless satisfactory substitutes can be made near at hand. Several coal tars and a number of Mexican oil field products remain to be tested.

2. The addition of acid shows no particular benefit. A slight alkalinity not exceeding 0.03 lb. lime per ton solution is satisfactory.

3. Heating the solution does not materially improve results.

4. The flotation tailing is cyanided readily, the presence of the small quantity of oil having no deleterious effect.

5. The raw flotation concentrate cyanides with no more difficulty than a gravity concentrate; the cyanide consumption maintains more or less the same ratio as in the crude ore, viz.: 4 g. sodium cyanide per gram of silver; a 98 per cent. extraction of gold and silver in the concentrate is obtainable.

6. In the small laboratory machines without cleaning cells, a recovery

by flotation of 65 to 70 per cent. can be made at a ratio of concentration of between 80 and 90 to 1.

7. In the 100-ton trial plant now under construction, equipped with middling returns and cleaning cell, it is expected that these results can be improved.

8. It is possible, though not certain, that flotation, even with this clean ore, may replace cyanidation to a large extent.

POWER

The Pachuca district is supplied with electric power generated by several hydro-electric plants owned by three power-distributing companies, practically under one control. The main plant, owned by the Mexican Light & Power Co. and located at Necaxa, Puebla, about 75 miles by transmission line to the east, also furnishes energy for Mexico City and El Oro. The 50-cycle alternating current is transmitted at 85,000 volts to the Pachuca substation where it is stepped down to 20,000 and 6,000 volts for local distribution. The power is sold at the substation of the purchaser on the basis of integrating wattmeter readings, no peak or installation loads being considered. A power factor of at least 80 to 85 per cent. is required of the purchaser and prices vary from $42.50 to $50 per horsepower-year, depending on amount consumed.

Although most of the mines are wet, there are no reserve steam plants in the district, with the exception of a relatively few old steam hoists that could be used for baling in an emergency. Nevertheless, the several widely separated hydro-electric plants with individual transmission lines provide a very reliable power supply.

GENERAL REMARKS

As can be imagined, operating conditions in Mexico during the past several years have been subjected to many unusual difficulties, although the Pachuca district perhaps has been as fortunate as any in maintaining a fair rate of work. This has been due principally to its advantageous geographical location, numerous railroad connections and steady power service. Added to these favorable factors, there has been coöperation among the mining companies and a considerable resourcefulness in looking after the labor and in bringing in supplies.

Like other countries at present under strain, Mexico has found it necessary to increase all taxes very materially, including those on mining property and on metals produced and exported. Very luckily, the prevailing high metal prices still enable the silver and base-metal producers to make a profit, but the gold-mine owner is hard hit with no rise in the market price of his product to offset steadily increasing costs as affected by the taxes, higher prices for supplies, etc.

The labor outlook is not encouraging. As a natural outcome of several years of revolutions, labor is also passing through a similar period of unrest, and much tact combined with fair and firm treatment will be necessary to meet this situation in any way successfully. The conditions are likely to be accentuated, due to a shortage of labor supply when mining activity throughout Mexico is again possible.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

A Comparative Test of The Marathon, Chilean and Hardinge Mills

BY F. C. BLICKENSDERFER,[*] MORENCI, ARIZ.

(Arizona Meeting, September 1916)

INTRODUCTION

DURING 1914 and 1915 extensive experiments were conducted at the concentrator of The Detroit Copper Mining Co. of Arizona, at Morenci, Ariz., in order to test the relative grinding efficiencies of the Marathon, the Chilean and the Hardinge Mills.

The Marathon mill used in the experiments was the first one ever built by the manufacturers; and since this was the first time that it was given a thorough tryout, with such remarkable results, the data have been assembled in the following paper and submitted in the interests of the milling profession.

DESCRIPTION OF THE FLOW SHEET

The concentrator flow sheet, as it existed at the time of these tests, is shown in Fig. 1. The total mill feed, as delivered from the crushing plant, had as its maximum size 1-in. material. This material was screened dry on Cole Zig-Zag screens fitted with 4-mesh openings; the oversize going to one pair of coarse rolls, the undersize to five Wilfley tables fitted with Butchart National riffles. The coarse-roll product was elevated and screened wet on Zig-Zag screens fitted with 4-mesh openings, the undersize going to the five tables mentioned above and the oversize to one pair of fine rolls.

The fine-roll product was elevated and screened together with the coarse-roll product; so that, before any concentrating operations took place, the total mill feed was reduced to pass a 4-mesh screen.

After concentration on Butchart-riffled tables, the tailings from these tables passed to de-sliming hoppers, where two products were formed, *viz.*, an overflow and a discharge. The discharge constituted the feed for the Chilean mills. It contained 16 per cent. slime (−200 mesh) and 3 parts of water to 1 of solids. The concentrator was equipped regularly with four Chilean mills (Monadnock type) to the unit; but, in order to provide similar conditions for a test, two of these mills were replaced by a Hardinge mill and a Marathon mill. The Hardinge mill was fed with the discharge from the de-sliming hopper; but a greater proportion of the slime was removed in order to get a drier feed, containing but 8 per cent. slime (−200 mesh) and with a ratio of 1.3 parts water to 1 of solids. The Marathon mill was also fed with the discharge from the de-sliming hopper, containing 12 per cent. slime and having a water to solids ratio of 1.8 : 1. Three shovel-wheels were temporarily installed to remove all of the slime from the table tails, for the purpose of giving a much larger tonnage to the Marathon mill, it being found that the total normal unit tonnage could be handled by the three shovel-wheels.

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[*] Concentrator Metallurgist, Detroit Copper Mining Co.

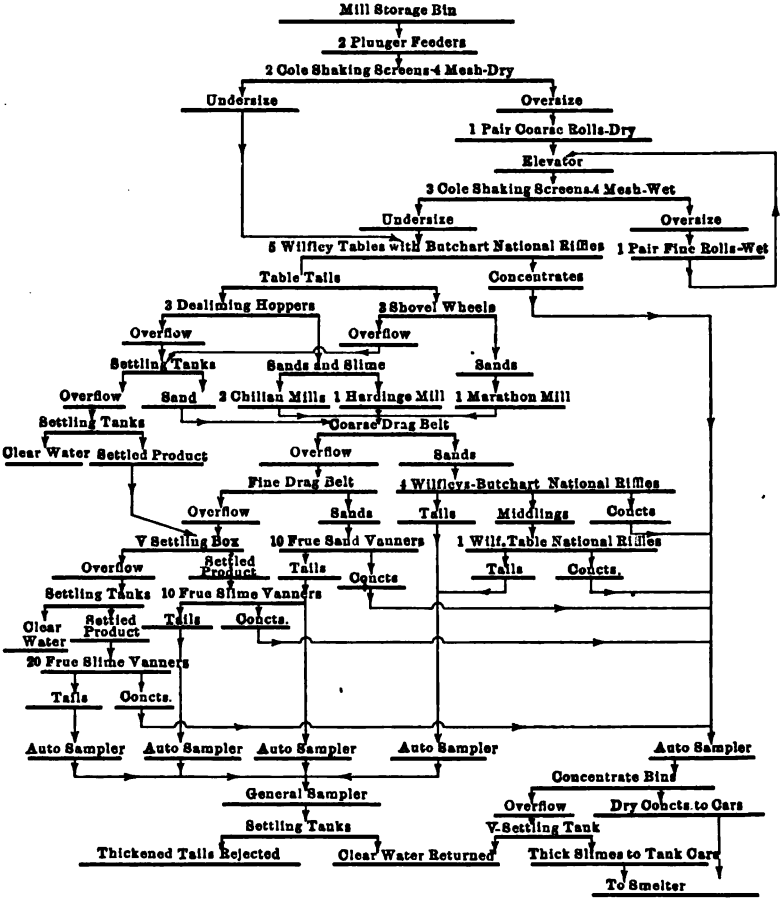

Fig. 1.—Flow Sheet of 750-Ton Concentrator Unit of Detroit Copper Mining Co., Morenci, Ariz.

All of the mill products passed to a coarse drag belt which took out the coarse sand for treatment on the Butchart-riffled tables; the coarse drag-belt overflow flowed to a fine drag belt, which removed fine sand for treatment on Frue sand-vanners. The overflow from the fine drag belt passed to a V-box which removed a thick product for treatment on Frue slime-vanners; the V-box overflow was thickened for final treatment on Frue slime-vanners. All rejected tailings were sampled separately as they came from each group of machines, then a general sample of the tails was taken after all of the tailings came together at a common point. The clear water at each point, if at a sufficiently high level, was used for wash water on machines below; but, if not, it was collected at the bottom of the concentrator and pumped to tanks at the top to be used over again.

The crude milling ore is a quartz-monzonite porphyry with chalcocite as the copper mineral. An average assay for one year is: Cu, 2.80; SiO_2, 59.5; Al_2O_3, 16.8; Fe, 5.0; S, 5.0; CaO, 0.7; and MgO, 0.9 per cent. The ore contains a small percentage of copper sulphate, which is soluble in water. This "acid water" is very destructive to all iron parts, so that iron consumed in all these tests is excessive to a small degree for quantities of feed crushed. Unslacked lime is added continuously at the rate of 2,000 lb. daily to circulating mill water to neutralize the copper sulphate.

GENERAL CONDITIONS REGARDING THE TESTS

These experiments were carried on for several months, so that results obtained are not spasmodic, but represent the average results of routine work of each mill under all the variable conditions encountered in actual milling practice.

The Chilean and Hardinge mills ran together for a continuous period of 63 days, or for the length of time required to wear out the tires and die of the Chilean mill. The Hardinge mill ran the first 30 days with an old lining. The old lining was then removed, a new one put in and the test completed.

The average results of the Hardinge and Chilean mills for the 63-day period will be used in comparison with the Marathon mill. Test No. 1 of the Marathon mill was run under the same conditions regarding feed as the Hardinge and Chilean, so that an excellent comparison is possible.

Some time elapsed between tests Nos. 1 and 2 of the Marathon mill, during which some preliminary tests were made to determine the best conditions for operating with a heavy tonnage. During this time the three shovel-wheels were installed to remove all slimes and the greater amount of water. Also about this time the tonnage per unit was reduced from 750 tons to 500 tons per 24 hr., and this total unit feed was sent through the Marathon mill. Subtracting the weight of concentrates produced in primary table concentration, and the weight of slimes removed by the shovel-wheels preceding the Marathon mill, a net dry weight of 430 tons per 24 hr. was obtained. The total recovery of the unit under these conditions was almost exactly the same as the second unit operating three Chilean mills in the same step. This fact paved the way for a heavy tonnage on the Marathon mill, known in this report as test No. 2.

Description of the Three Types of Mills Tested

The Chilean mill used in the test was of the Monadnock type, 5 ft. in diameter, and fitted with 1½-mm. round-hole punched screens. The Chilean die (or ring), when new, weighed 1,722 lb., the three tires together weighed 2,526 lb. A 50-hp. motor furnished power to drive the Chilean mill, transmission taking place in two stages, viz., from motor pulley to line shaft, thence to Chilean mill belt wheel.

The cylindrical portion of the Hardinge mill was 8 ft. in diameter by 3 ft. long. It was lined with Danish pebbles of uniform size, set in concrete. The method of lining was as follows: The lower half was lined first and allowed to "set," then the mill was turned over so that the unlined portion was below. This section was then lined and the total lining allowed to set 48 hr. The Hardinge mill was charged with 5 tons of Danish pebbles of fairly uniform size. Power was furnished by a 75-hp. motor direct-connected to a gear shaft which in turn engaged with the large gear wheel of the same diameter as the mill itself.

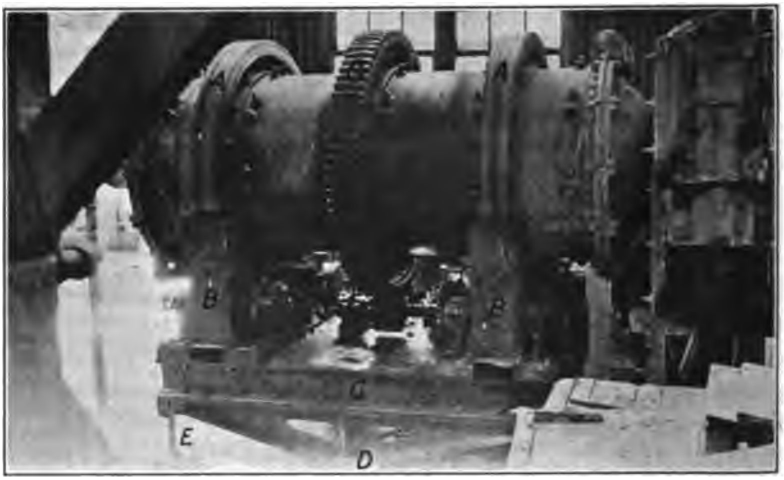

Fig. 2.—Old Marathon Mill.

The Marathon mill consisted essentially of a tube, 3 ft. in diameter by 7 ft. long, surrounded by two tires, *A* (Fig. 2), near each end, these tires being supported by and revolving upon two pairs of rollers, *B*. Two idler rollers, *H*, keep the tires in alignment with the rollers when the mill is in action. The four rollers *B* are set on frame *C* and are rigidly bolted to it. Frame *C* is pivoted at its center, *D*, thus providing a tilting arrangement, which permits of adjustment to any desired slope by bolts, *E*, at either end of the frame. The ends are bolted to the mill tube, and can be quickly and easily removed for the inspection or renewal of crushing parts. The feeding arrangement consists of an L-shaped tube, the horizontal leg of which discharges into the mill through a circular opening in the center of the head end. The product, in wet crushing, is discharged through a 6-in. pipe centrally located in the tail end. In dry crushing, the Marathon mill has a special end with slots in the periphery for the free discharge of the crushed product. A large hopper is set below the discharge end to collect the product.

The Marathon mill lining consists of 16 cast-iron plates bolted to the mill shell, the total weight of the plates being 4,480 lb. The crushing medium is composed of iron rods ranging in size from ½ in. to 2 in. in diameter; the initial charge of rods weighed 7,000 lb. The mill is driven by a gear wheel, *G*, connected by two sets of gears to a drive wheel, which is belted to a 25-hp. motor.

Modifications of Original Design of Marathon Mill

During the progress of the experimental work with the Marathon mill, several construction defects developed, which had to be remedied before further operation could proceed. A preliminary run with no load of ore in the mill, showed that the understructure was not rigid enough to prevent excessive vibration. The top and bottom of the frame were entirely covered with ½-in. and 5⁄16-in. sheet-steel plates respectively, the plates being firmly riveted to the legs of channel irons composing the top of the tilting frame.

The feed spout as furnished with the mill gave endless trouble with a wet feed, because the pulp was dashed out around the feed spout where it enters the center of the mill. The feed spout was replaced by a scoop feeder which was bolted to the head end of the mill.

A circular cast-iron liner (1¼ in. thick at the edges by 2¼ in. at the center) of the same diameter as the inside of the mill, was bolted to the head end to protect it against excessive wear.

The roller bearings supporting the axles of rollers were very unsatisfactory. Being of a spiral type, they were forced to extend by the weight of the mill to such a degree that the bearing end was thrust off. This extension of spiral rollers caused a reduction in their diameter, which resulted in a loose wabbly bearing. In case of such a failure the roller-bearing parts were renewed.

Methods of Operating the Three Types of Mills

The best method of operating the Chilean mill is perhaps too well known to require a detailed description. It is essential to have sufficient water in the feed for the Chilean mill to obtain best results from the mill screens, for if the pulp is too thick the screens clog and cause the mill to become overloaded. If too great a quantity of pulp is fed to the mill, overloading follows at once and the mill does practically no grinding. When the Chilean mill became overloaded, the power required to drive it was about one-third more than under normal conditions. An average ratio of 3 parts water to 1 part solids (25 per cent. solids) was the best pulp to feed to the Chilean mill for all-around good work. The speed was 40 r.p.m.

The Hardinge required a less dilute pulp (42.8 per cent. solids) for best results. It is practically impossible to overload the Hardinge mill, "so to speak," as long as the mill is able to discharge the product, but constant attention is required to prevent "oversize." When its grinding capacity is exceeded, a considerable amount of very coarse material traverses the length of the mill without receiving a single blow from the falling pebbles. Special attention was given each of the mills in order to keep them up to full capacity with the production of a minimum quantity of "oversize." The mill was operated at 29 r.p.m.

The Marathon mill in Test No. 1 was operated as a fine grinding machine so as to give a product of the same fineness as the Hardinge and the Chilean mills. The most favorable results were obtained with pulp that contained 36.7 per cent. solids. When the Marathon mill became overloaded it was surprising to find no coarse oversize in the product, *but a product that was uniformly coarser.* This is a fact that requires special notice. It is equivalent to saying that if the mill is once started up it is not necessary to pay any attention to the product, for if overloaded, the whole product will be slightly coarser, and if underloaded the product will be slightly finer.

In test No. 2 of the Marathon mill the amount of water and slime in the feed was reduced to a minimum by shovel-wheels to secure a less volume of pulp and a consequent slower velocity of pulp through the mill. The tonnage of test No. 2 was greatly increased over that of test No. 1, and the Hardinge and Chilean mill tonnages. The feed contained an average of 63.5 per cent. solids, and only 2.92 per cent. slime (−200 mesh). The speed in all the tests of the Marathon mill was 30 r.p.m. In a diameter of 3 ft. at 30 r.p.m. there is no dead zone in the periphery of the mill, while excessive slipping of rods is prevented by having the liner plates thicker at each edge than in the middle, resulting in a corrugated interior surface. The lining retains this corrugated surface until it is worn out. The rods wear evenly from end to end, do not get crosswise in the mill and are reduced to a diameter of ⅜ in. before they begin to crush, flatten out or break into pieces. When worn to this thinness, some rods roll up and are discharged; the short half-length pieces retain their positions in the charge of rods without any serious results, but may reduce crushing efficiency to some extent. It is good practice to take out the entire charge of rods every 10 days, sort out the small and "disabled" rods and replace the loss in weight with 1¼-in. rods. As soon as the weight of rods consumed per day was determined, their loss in weight was replaced daily with fresh rods. The mill must be closed down to add the daily charge of rods. During these tests the Marathon mill was tilted ¼ in. per foot, but later tests proved that it would produce equally as good results when run horizontally.

Some experiments were made on coarse (+ ½-in.) material both wet and dry, using the peripheral discharge end and feed spout. The charge of rods in the mill was increased by the addition of steel shafting 4 in. in diameter. These tests were not extended over any length of time, nor were sufficiently conclusive results obtained to warrant their mention. The primary fact established was this: The mill was able to handle coarse feed up to 2 in. in a way to encourage further experiment.

Explanation of the Action of the Marathon Mill

It might be well at this point to offer an explanation of the action that takes place within the Marathon mill.

For the sake of comparison, if we select any type of mill with spherical-shaped grinding media, the Hardinge mill for example, and consider a single crushing element, we have a row of pebbles in contact with another row of pebbles as indicated in Fig. 3.

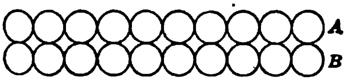

FIG. 3.

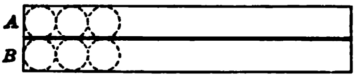

FIG. 4.

Now when falling, row *A* strikes row *B*, or if rolling together, row *A* rolls on row *B*. It is evident that crushing takes place where one pebble strikes or rolls against another, or at their point of contact. The number of points of contact in an element is determined by the size of pebbles; the smaller the size of pebbles, the greater the number of points of contact and *vice versa*. But it has been demonstrated in actual practice that the pebbles must be above a certain size to develop a crushing blow when falling of sufficient force to be of any value. Hence crushing ability depends not only on points of contact, but also on the size of pebbles and on their mass (by *mass* is meant specific gravity).

Since the number of points of contact must vary inversely as the size of pebbles, the two factors are compromised by using various sizes of pebbles. In order to take advantage of the mass factor, iron balls have been recently substituted for pebbles as a crushing medium.

Now consider a solid bar for the grinding medium, in the case of the Marathon mill, as shown in Fig. 4. When falling, rod *A* strikes rod *B*, or if rolling together, rod *A* rolls on rod *B*. There is a continuous row of points of contact, or geometrically speaking, a line. It has been shown in experiments that the rods remain parallel when the mill is in operation. Hence we have the maximum number of contact points that can exist between the surfaces of two rods (cylindrically shaped). Again, we may consider the two rods *A* and *B* of Fig. 4 as made up of solid iron balls packed closely and bound together throughout their length by additional iron which is 0.4764 of their own weight. Therefore, the crushing ability of iron rods—bearing in mind that crushing ability depends upon points of contact and upon the mass of crushing medium—must be greatly superior to those mills which use spherical-shaped crushing media. The relatively small diameter of rods, 2 in. to 3/8 in., causes them to fit compactly, thus requiring a minimum falling distance to deliver a crushing blow. The center of mass of a charge of rods is much nearer the axis of the mill than in a mill of greater diameter whose crushing charge consists of steel balls or pebbles.

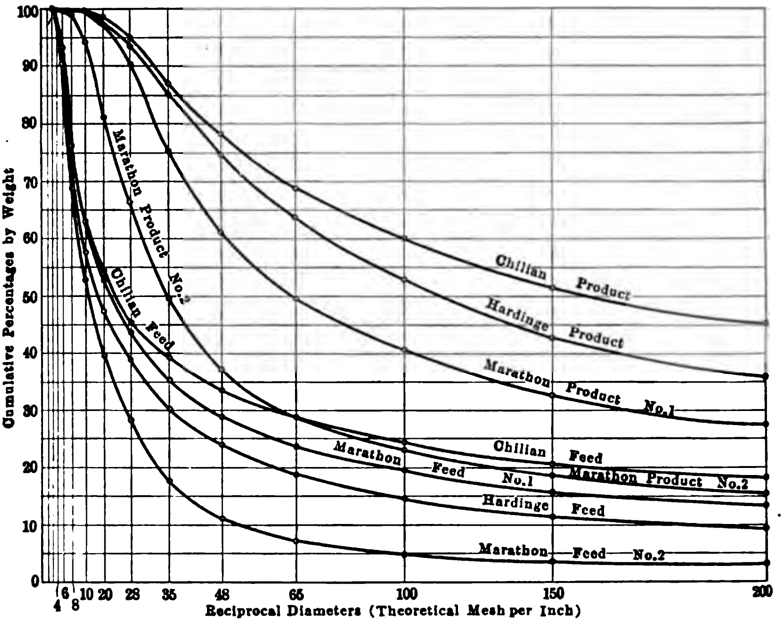

FIG. 5.—DIAGRAM OF CRUSHING EFFICIENCY—GATES METHOD.

It is necessary in ball and pebble mills to have a sufficient weight of charge to deliver a crushing blow. This condition is effected either by making the mill of large diameter or by increasing its length. When the length is increased the power factor increases, when the diameter is increased the power factor increases very rapidly and at the same time there is danger of the peripheral speed becoming too great, tending to form a dead zone.

TABLES OF RESULTS SHOWING ITEMS AND CONCLUSIONS

TABLE 1.—*Cost Comparisons*

	Marathon Test No. 1, Per Cent.	Marathon Test No. 2, Per Cent.	Hardinge Mill, Per Cent.	Chilean Mill, Per Cent.
Operating Labor	13.38	11.21	7.75	11.63
Labor on Repairs	12.20	12.27	3.85	7.43
Material on Repairs	40.38	42.29	28.40	22.52
Power	34.04	34.23	60.00	58.42
	100.00	100.00	100.00	100.00

Marathon No. 1 is Exceeded in Operating Costs by	Excess Cost, Per Cent.
Marathon No. 2	19.34
Hardinge	72.56
Chilean	15.00

Chilean is Exceeded in Operating Costs by	Excess Cost, Per Cent.
Marathon No 2	3.77
Hardinge	50.04

Marathon No. 2 is Exceeded in Operating Cost by	Excess Cost, Per Cent.
Hardinge	44.59

Costs are generally governed by local conditions. In each of the divisions, Operating Labor, Labor on Repairs, Material on Repairs, and Power, the percentage of the total cost of the mill is represented, no correction being made for the tonnage. From the second part of this table it is to be noted that the Marathon mill costs as operated in test No. 1 are the least; the Chilean mill costs are the next lowest; then the Marathon mill costs as operated in test No. 2; and the Hardinge mill costs are the highest. In comparing the costs of one mill with those of any other, for instance the Hardinge costs and the Chilean costs, the mill with the lowest costs is taken as 100 per cent. The Hardinge costs on this basis are 150.04 per cent., or 50.04 per cent. in excess of the Chilean costs.

Material on Repairs includes rods and liners for the Marathon mill, die and tires for the Chilean mill, and pebble lining and pebbles for the Hardinge mill.

In each case the power factor is strongly in favor of the Marathon mill. The tonnage factor is omitted at this point because it properly belongs to a comparison of grinding efficiencies.

TABLE 2.—*Record of the Average for Each Test*

Type of Mill	Running Time in Per Cent. of Total Time	Time Lost, Per Cent.	Total Actual Running Time in Hours	Tons Dry Feed in 24 Hr.	Tons Dry Feed per Hour	Tons Pulp in 24 Hr.	Tons Pulp per Hour	Per Cent. Solids in Feed
Marathon No. 1........	94.00	6.00	383.0	236.5	9.854	662.4	27.60	35.7
Marathon No. 2........	94.25	5.75	200.0	440.0	18.333	693.0	28.87	63.5
Hardinge.............	96.25	3.75	1,455.3	244.0	10.166	570.1	23.75	42.8
Chilean...............	93.75	6.25	1,401.7	237.0	9.875	951.4	39.64	24.9

Type of Mill	Ratio of Solids to Water	Water with Each Ton of Feed, Gallons	Water with 24-Hr. Tonnage, Gallons	Water in Pulp, Gallons per Minute	Kilowatts Consumed per Hour	Horsepower Consumed per Hour	Tons Dry Feed per Hp.-hr.	Tons Dry Feed per Hp.-day	Speed of Mills, R.p.m.
Mar. No. 1.....	1 to 1.8	432.2	102,215	70.98	13.880	18.60	0.5316	12.758	30.27
Mar. No. 2.....	1 to 0.57	138.0	60,720	42.17	16.791	22.50	0.8148	19.555	30.10
Hardinge.....	1 to 1.3	320.6	78,226	54.32	42.208	56.56	0.1797	4.313	29.03
Chilean.......	1 to 3.0	723.8	171,540	119.13	27.388	36.70	0.2691	6.485	39.88

The term pulp is here used to signify a mixture of water and dry feed, and the quantity of water in the pulp is the amount in combination with the dry feed.

The delays of the Hardinge mill are due to relining and allowing time for lining to "set." The delays of Chilean mill are due to renewing of crushing parts, replacing of screens and oiling of mill. The delays of the Marathon mill are due to changing of rod charge, addition of new rods, relining, and extraordinary repairs on defective parts.

The power consumed by the Marathon mill is practically constant, the rise in power consumed in test No. 2 being due to an increased charge of rods. The Hardinge power is also constant. The Chilean mill consumes less power when the tires and die are new than when they have been in operation for some time. If mills are slightly overloaded, the power increases very rapidly.

TABLE 3.—*Average of Test Conditions*

Type of Mill	Stationary Grinding Parts					Movable Grinding Parts					Total Grinding Parts
	Kind of Stationary Grinding Surface	Wt. of Stationary Grinding Surface, Pounds	Life of Liners or Die, Days	Liners or Die Consumed in 24 Hr., Lb.	Liners or Die per Ton, Pounds	Charge Used in Mill	Wt. of Initial Charge, Pounds	Total Pounds Added During Test	Pounds Consumed in 24 Hr.	Consumed per Ton of Feed, Pounds	Consumed Per Ton of Feed, Pounds
Mar. No. 1...	Iron plates	4,480	82	54.707	0.2313	Iron rods	6,395	2,404	150.64	0.63697	0.86827
Mar. No. 2...		4,480	72	62.256	0.14149		7,124	1,441	172.92	0.39295	0.53544
Hardinge.....	Pebbles	6,740	159	41.761	0.17115	Pebbles	10,175	31,956	527.00	2.16	2.33115
Chilean......	Die	1,722	63	27.333	0.11533	3 Tires	2,526	None	41.66	0.1762	0.29153

This table expresses the life of linings, the rate of wear, and pounds of each grinding medium sacrificed per ton of ore ground. The Chilean mill die corresponds to liners and lining of the other two mills. It is to be noted that the Chilean mill required the least amount of grinding medium per ton. It is the auxiliary parts, such as screens, screen frames, tire wheels or mullers, feed spouts, ploughs, mortars, and the power, that completely reverse this good quality.

TABLE 4.—*Comparison of Grinding Efficiencies*

Marathon No. 1						Marathon No. 2					
Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product	Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product
+4	4.46	0.13	0.58	......	....	+4	4.46	0.09	0.40	......	....
+6	6.33	6.49	41.08	......	....	+6	6.33	9.37	59.31	0.04	0.25
+8	8.93	17.12	152.88	0.01	0.09	+8	8.93	21.58	192.71	0.81	7.23
+10	12.66	13.40	169.64	0.28	3.54	+10	12.66	15.90	201.29	4.90	62.03
+14	17.86	10.44	186.46	2.12	37.86	+14	17.86	13.55	242.00	12.05	215.21
+20	25.38	8.51	215.98	7.06	179.18	+20	25.38	11.14	282.73	15.55	394.66
+28	35.71	8.35	298.18	14.77	527.44	+28	35.71	10.42	372.10	16.82	600.64
+35	50.51	6.61	333.87	14.43	728.86	+35	50.51	6.96	351.55	12.50	631.37
+48	71.43	5.30	378.58	11.68	834.30	+48	71.43	3.92	280.01	8.53	609.30
+65	101.01	3.91	394.95	8.72	880.81	+65	101.01	1.97	198.99	5.43	548.48
+100	142.86	3.78	540.01	8.05	1,150.02	+100	142.86	1.15	164.29	4.67	667.16
+150	200.00	2.46	492.00	5.15	1,030.00	+150	200.00	0.66	132.00	2.82	564.00
+200	285.71	1.29	368.57	2.67	762.85	+200	285.71	0.37	105.71	1.70	485.71
−200	454.55	12.21	5,550.05	25.06	11,391.02	−200	454.55	2.92	1,327.29	14.18	6,445.52
Total...	......	100.00	9,122.83	100.00	17,525.97	Total	......	100.00	3,910.38	100.00	11,231.56

TABLE. 4.—(*Continued*)

Hardinge Mill						Chilean Mill					
Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product	Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product
+4	4.46	0.97	4.33	......	....	+4	4.46	0.94	4.19	......	....
+6	6.33	8.43	53.36	......	....	+6	6.33	7.71	48.80	......	....
+8	8.93	18.79	167.79	0.07	0.63	+8	8.93	16.49	147.26	0.05	0.45
+10	12.66	13.86	175.47	0.58	7.34	+10	12.66	11.84	149.89	0.10	1.26
+14	17.86	10.45	186.64	1.95	34.83	+14	17.86	8.98	160.38	1.07	19.11
+20	25.38	8.49	215.48	3.80	96.44	+20	25.38	7.18	182.23	3.63	92.13
+28	35.71	8.40	299.96	8.17	291.75	+28	35.71	7.43	265.33	7.75	276.75
+35	50.51	6.53	329.83	10.45	527.83	+35	50.51	6.04	305.08	9.08	458.63
+48	71.43	5.29	377.86	11.60	828.59	+48	71.43	4.80	342.86	9.38	670.01
+65	101.01	4.10	414.14	10.25	1,035.35	+65	101.01	3.89	392.93	8.53	861.62
+100	142.86	3.47	495.72	10.40	1,485.74	+100	142.86	3.76	537.15	8.95	1,278.60
+150	200.00	1.81	362.00	6.75	1,350.00	+150	200.00	2.58	516.00	6.23	1,246.00
+200	285.71	1.04	297.14	4.13	1,179.98	+200	285.71	1.82	519.99	4.21	1,202.84
−200	454.55	8.37	3,804.58	31.85	14,477.42	−200	454.55	16.54	7,518.26	41.02	18,645.64
Total....	......	100.00	7,184.30	100.00	21,315.90	Total		100.00	11,090.35	100.00	24,753.04

	Marathon No. 1	Marathon No. 2	Hardinge	Chilean
Work units in product.........	17,525.97	11,231.56	21,315.90	24,753.04
Work units in feed	9,122.83	3,910.38	7,184.30	11,090.35
Work units expended in crushing	8,403.14	7,321.18	14,131.60	13,662.69
Work units expended in crushing corrected for tonnage..........	107,207.3	143,165.70	60,949.60	88,602.54

Excess Efficiency of Marathon No. 2 over	Per Cent.
Marathon No. 1	33.54
Hardinge..	134.80
Chilean	61.58

Excess Efficiency of Marathon No. 1 over	Per Cent.
Hardinge..	75.90
Chilean	30.52

Excess Efficiency of Chilean over	Per Cent.
Hardinge..	45.37

Tyler Standard screens were used in all of the screen analyses, and constants of this screen scale are employed in computations. The principle upon which the comparison of efficiencies is based may be summed up in a few words: "Work done in crushing is proportional to surface exposed in crushing, and therefore nearly proportional to reduction in diameter," or, "nearly proportional to the reciprocals of diameters crushed to." This is the well-known and substantial efficiency comparison of Del Mar. Per cent. weight is multiplied by reciprocals of *average diameter* in both feed and product, the difference in these two quantities measuring the work expended in crushing. When this quantity is multiplied by tons per horsepower-day, we involve the two factors that complete the efficiency comparison. This method is commendable with but one exception; the assuming of a value for the average diameter of −200-mesh material. This assumed value probably comes somewhere near to the approximate value, but no more. In logarithmic plotting of feeds and products, Fig. 6, the lines have been extended in a smooth curve to the edge of the paper, and in every case the assumed value is only approximate. It is of no advantage to assume this value, especially when there is large quantity of −200-mesh material. The curve of Chilean product lacks sufficient points to tell just where it does extend. The others have a fair degree of accuracy.

Crushing-Surface Diagram

This is the method proposed by Arthur O. Gates for comparing efficiencies in grinding. Cumulative percentages are plotted vertically and reciprocals of screen-mesh diameters, horizontally (Fig. 5). This

TABLE 5.—*Gates Method of Comparison*

Mesh	Reciprocal of Mesh Diameter	Mar. No. 1 Feed Cumulative, Per Cent. by Weight	Mar. No. 1 Product Cumulative, Per Cent. by Weight	Mar. No. 2 Feed Cumulative, Per Cent. by Weight	Mar. No. 2 Product Cumulative, Per Cent. by Weight	Hardinge Feed Cumulative, Per Cent. by Weight	Hardinge Product Cumulative, Per Cent. by Weight	Chilean Feed Cumulative, Per Cent. by Weight	Chilean Product Cumulative, Per Cent. by Weight
+4	5.40	100.00	....	100.00	....	100.00	....	100.00	....
+6	7.63	99.87	....	99.91	100.00	99.03	....	99.06	....
8	10.75	93.38	100.00	90.54	99.96	90.60	100.00	91.35	100.00
10	15.38	76.26	99.99	68.96	99.15	71.81	99.93	74.86	99.95
14	21.74	62.86	99.71	53.06	94.25	57.95	99.35	63.02	99.85
20	30.48	52.42	97.59	39.51	82.20	47.50	97.40	54.04	98.78
28	43.10	43.91	90.53	28.37	66.65	39.01	93.60	46.86	95.15
35	60.98	35.56	75.76	17.95	49.83	30.61	85.43	39.43	87.40
48	86.20	28.95	61.33	10.99	37.33	24.08	74.98	33.39	78.32
65	121.95	23.65	49.65	7.07	28.80	18.79	63.38	28.59	68.94
100	172.41	19.74	49.93	5.10	23.37	14.69	53.13	24.70	60.41
150	243.90	15.96	32.88	3.95	18.70	11.22	42.73	20.90	51.46
200	344.82	13.50	27.73	3.29	15.88	9.41	35.98	18.36	45.23
−200	....	12.21	25.06	2.92	14.18	8.37	31.85	16.54	41.02

TABLE 5.—(*Continued*)

From Crushing Diagram by Gates Method (Fig. 5)

	Square Inches	Efficiency Corrected for Tonnage
Marathon No. 1	7.51	95.812
Marathon No. 2	6.65	130.004
Hardinge	12.02	51.842
Chilean	11.12	72.113

Excess Efficiency of Marathon No. 2 over	Per Cent.
Marathon No. 1	35.69
Hardinge	150.75
Chilean	80.29
Excess Efficiency of Marathon No. 1 over	**Per Cent.**
Hardinge	84.81
Chilean	32.86
Excess Efficiency of Chilean over	**Per Cent.**
Hardinge	39.10

diagram averages the diameters without calculation, and areas upon it are proportional to surface produced, and, in accordance with Rittinger's law, to energy spent in crushing alone. Measuring these areas up to 200 mesh and multiplying by tons per horsepower-day, we obtain efficiency units for comparison, as indicated in Table 5.

TABLE 6.—*Comparison of Del Mar and Gates Methods*

Excess Units by	Del Mar, Per Cent.	Gates, Per Cent.	Numerical Average, Per Cent.
Marathon No. 2 over			
Marathon No 1	33.54	35.69	34.61
Hardinge	134.80	150.75	142.77
Chilean	61.58	80.29	70.93
Marathon No. 1 over			
Hardinge	75.90	84.81	80.35
Chilean	30.52	32.86	31.69
Chilean over			
Hardinge	45.37	39.10	42.23

In view of the fact that in Del Mar's system the −200 mesh material is taken into account and in the other it is not, we may consider a check existing between the two methods with but one exception, comparing

Marathon No. 2 with Chilean, the two extremes of feeds and products; the one producing the most slime and the other producing the least slime. To be impartial and conservative at the same time, we will adopt the numerical average of the two methods.

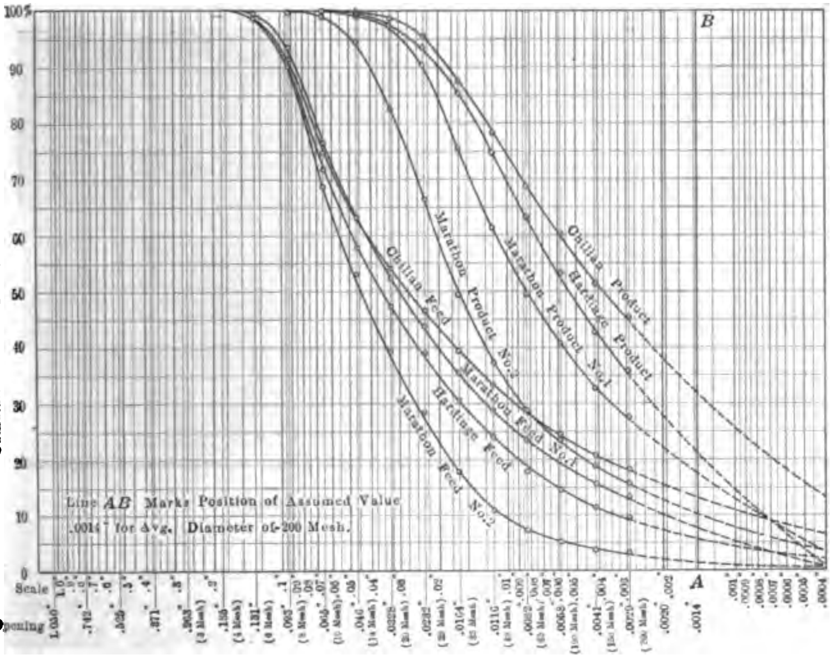

FIG. 6.—CUMULATIVE LOGARITHMIC PLOT OF SCREEN ANALYSES ON FEEDS AND PRODUCTS.

SUMMARY

For the material treated and under the conditions of these tests, the Marathon mill is far superior to the Hardinge and Chilean mills in grinding efficiency. Of the Hardinge and Chilean mills, the Chilean is superior to the Hardinge. The Marathon mill will bear further investigation and experiment. Its costs will be much less in the actual installation of several mills than in a single mill requiring perfection of parts, as did this one.

On the basis of the results obtained in these tests, two mills, 4 ft. in diameter by 8 ft. long, for both coarse and fine grinding (Fig. 7) have been ordered, the principal features of which are as follows:

(*a*) Feed scoop attached to the head end of the mill. There are three channels in this scoop, entrance of each channel having a rectangular area of 108 sq. in. (9 by 12 in.).

(*b*) Sectional liners for head end and solid circular liner for tail end.

(*c*) Rigid base, with no tilting arrangement.

(*d*) Solid roller bearings (not spiral).

(*e*) Driving gear attached to tail end of shell, the pulley being supported rigidly by having its bearings set solidly on a separate concrete base. One of these mills has been in operation for a few days and is running smoothly after having considerable trouble with a hot bearing. These bearings are not able to stand up under such service. Babbitted bearings will undoubtedly remedy this trouble.

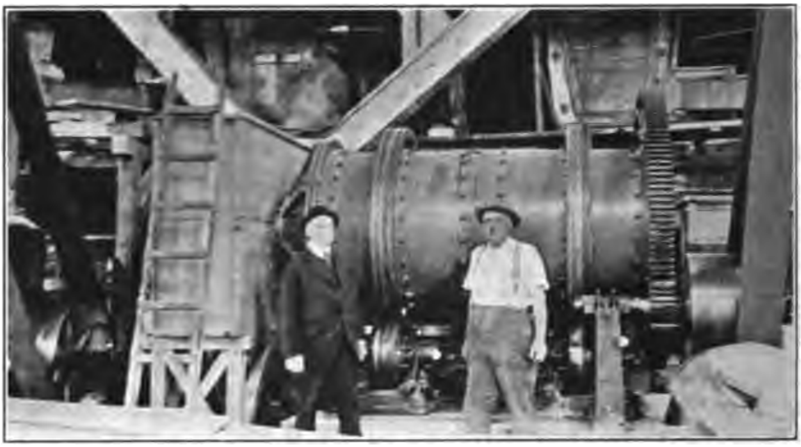

FIG. 7.—NEW MARATHON MILL.

Thanks are due to Milton H. McLean, General Manager of the Detroit Copper Mining Co. of Arizona, for permission to assemble and publish the above data.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Illuminating Power of Safety Lamps

BY W. M. WEIGEL,[*] E. M., STATE COLLEGE, PA.

(Arizona Meeting, September, 1916)

WHILE electric lamps both of the cap and hand type are being introduced into many mines requiring the use of safety lamps, the oil-burning safety lamp is still used in the great majority of cases, and even where the former are used the latter must continue to be employed when testing for gas. It is not the intention to discuss here the relative merits of the two types, but simply to give a comparison of the illuminating effect or candlepower of several kinds of ordinary safety lamps used for general work and for testing. Few data have been published in recent years on this subject and because of this it is hoped that the following information may be of some interest and value.

The lamps tested were in all cases of standard size and in good condition, all practically new, and were selected from a collection of about 100 with a view to obtaining results on as many different common types as possible. With the exception of the gasoline-burning lamps, five different oils were used in each lamp to determine whether particular oils were suited to particular lamps and whether there was any great difference in the illuminating power of the different oils, or whether some oils would maintain their illuminating power better than others throughout a working period of 8 hr. No attempt was made to determine the amount or cost of oil burned by each lamp, but the condition of the lamp, wick, glass, and gauzes was noted after each test.

Description of Apparatus Used

The candlepower was determined with a Weber photometer, Fig. 1, which consists of a tube *A* containing a movable milk-glass screen adjustable by the milled screw *B*. The distance of this screen from the standard light at *C* is read on the scale attached to *A*. The standard light consists of a benzine lamp so arranged that the height of the flame may be adjusted exactly by observing it against a scale in the lamp box through an opening in the front. The lamp to be tested is placed at *D* so as to illuminate a milk-glass screen in the holder at *E*. At the intersection of the axes of tube A and sight-tube F are placed two 45° prisms with their surfaces in contact so that the light from D may pass through without deflection while that from C is reflected at the contact in a direction parallel to that from D. The faces of the prisms are blackened around an elliptical field and on this field is an annular silvered ring which reflects the light from C. When viewed through the telescopic eyepiece, G, focused on the prisms, on outer annular ring and inner elliptical surface are seen illuminated by light D and an inner ring illuminated by that from C. With D in a fixed position, the milk-glass screen in A is adjusted until an evenly illuminated field is observed through the eyepiece. Then the screens in A and E are illuminated to the same intensity and the intensities of the standard and unknown light are to each other as the squares of their distances from the respective screens, or expressed as a formula

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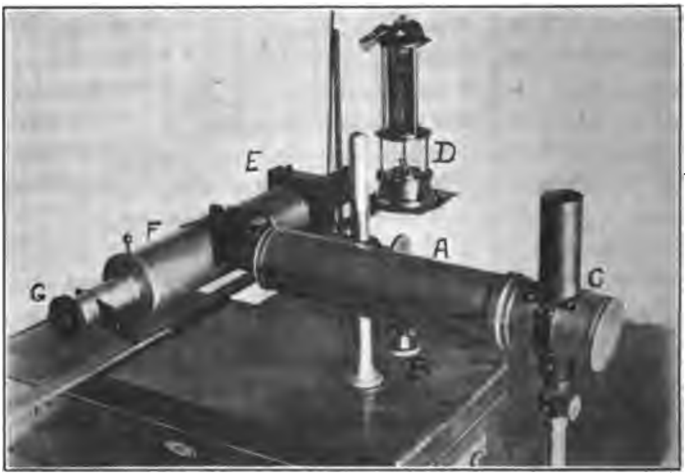

Fig. 1.—Weber Photometer Used in Determining Candlepower of Lamps.

$$i = \frac{R^2}{r^2}C$$

in which i = candlepower or intensity of light being tested,
R = distance of D (light being tested) from screen E,
r = distance of C (standard light) from screen in tube A,
C = constant involving the candlepower of the standard lamp, relative opacity of the two screens and characteristics of the apparatus.

Determination of the constant C was accomplished by placing a standard candle at D. The value of C is then given by writing the formula

$$C = \frac{r^2}{R^2}$$

as the intensity i is then unity. About 50 readings were taken using standard spermaceti candles. Several candles were employed and their illuminating power checked by weighing before and after use and determining the consumption of spermaceti per hour. A standard candle is supposed to burn 120 grains per hour. The candles used consumed from 123 to 124 grains per hour equivalent to a candlepower of 1.03. The average value obtained of the constant C was 0.36. The candle was placed at the same distance as the lamps tested and also at greater and less distances.

The specific gravity of the oils was determined with a Westphal balance, and the Baumé degrees calculated.

Description of the Lamps

The lamps tested were selected to give a wide range in variety and were of standard make and size (Figs. 2, 3 and 4). Before beginning each test with the several oils they were all thoroughly cleaned, the oil vessels rinsed out with gasoline and dried and fresh wicks put in. The shape and size of wicks, number and dimensions of gauzes and oils used are given in Table 1.

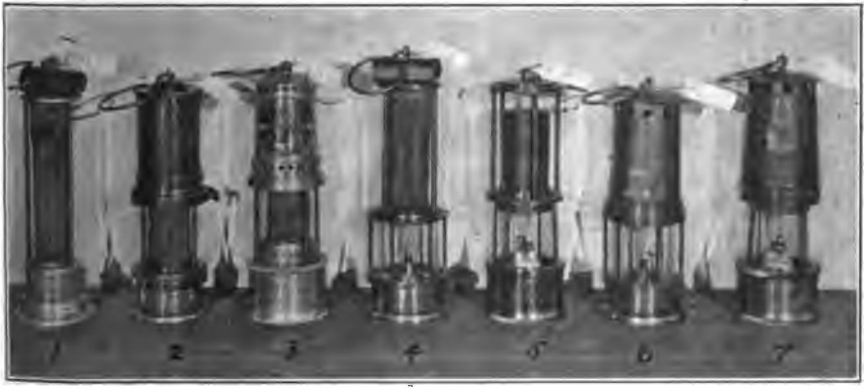

FIG. 2.—SAFETY LAMPS NOS. 1 TO 7 TESTED FOR ILLUMINATING POWER.

Lamp No. 1 is an ordinary Davy with aluminum frame.

Lamp No. 2 is a Davy lamp with bonnet and glass shield surrounding the lower portion of the gauze but which could be lifted when testing. It is also known as the Hughes-Evan Thomas lamp. Normally it takes its air through openings in the base ring covered on the inside by the lower end of the gauze.

Lamp No. 3 is of the same type but with aluminum oil vessel, frame, and bonnet. Also it has a Hailwood porcelain flat wick burner.

TABLE 1.—*Candlepower of Safety Lamps*

Number	Type	Location of Air Inlet	No. of Gauzes	Size of Gauzes, Inches	Bonnet	Size of Wick Tube, Inches R = round F = flat	Kind of Oil	Candlepower, Freshly Filled and Clean	Candlepower at End of 4 Hr.	Candlepower at End of 8 Hr.	Average Candlepower	Condition of Lamp at End of 8 Hr.
1	Common-Davy...	Entire gauze unprotected.	1	1½×5½	No	¼ R	Eureka..........	0.12	0.16	0.12	0.13	Gauze fairly clean, wick in good shape.
							Mineral Seal	0.10	0.09	0.11	0.10	Top of gauze smoked.
							Sperm..........	0.15	0.11	0.14	0.13	Required frequent attention and snuffing.
							Lard and kerosene	0.14	0.13	0.10	0.12	Good.
2	Hughes-Evan Thomas.	Below glass shield....	1	1½×5⅜	Yes	¼ R	Eureka..........	0.12	0.11	0.11	0.11	Same as No. 1.
							Mineral Seal	0.12	0.09	0.11	0.11	Top of gauze and cap smoked.
							Sperm..........	0.10	0.11	0.13	0.11	Required frequent attention and snuffing to keep burning.
							Lard and kerosene	0.14	0.10	0.12	0.12	Good.
3	Hughes-Evan Thomas.	Below glass shield....	1	1⅜ top 1½ bottom ×4½	Yes	3/32×½ F	Eureka..........	0.16	0.18	0.15	0.16	Same as No. 1.
							Mineral Seal	0.16	0.11	0.12	0.13	Gauze clean.
							Sperm..........	0.15	0.15	0.18	0.16	Same as No. 2, gauze clean.
							Lard and kerosene	0.17	0.09	0.13	0.13	Good.
4	Clanny..........	Base of gauze.......	1	1⅝×4¼	No	9/32 R	Eureka..........	0.34	0.35	0.43	0.37	Same as No. 1.
							Mineral Seal	0.32	0.30	0.33	0.32	Gauze clean.
							Sperm..........	0.27	0.24	0.25	0.25	Same as No. 3.
							Lard and kerosene	0.47	0.41	0.33	0.40	Good.
5	Clanny, but with perforated sheet copper gauze.	Base of gauze.......	1	1⅞×3	No	3/32×½ F	Eureka..........	0.34	0.42	0.33	0.36	Same as No. 1.
							Mineral Seal	0.42	0.35	0.36	0.38	Gauze clean.
							Sperm..........	0.37	0.28	0.37	0.34	Same as No. 3.
							Lard and kerosene	0.48	0.28	0.25	0.34	Good.
6	Evan Thomas No. 7.	Base of bonnet......	1	1 9/16 top 1 15/16 bottom ×3¼	Yes	9/32 R	Eureka..........	0.38	0.38	0.31	0.36	Same as No. 1.
							Mineral Seal	0.38	0.36	0.40	0.38	Top of gauze and cap slightly choked.
							Sperm..........	0.30	0.21	0.32	0.28	Same as No. 3.
							Lard and kerosene	0.46	0.41	0.34	0.40	Good.
7	Marsaut.........	Base of bonnet......	2	1 13/16 & 1 15/16×3½ 1 9/16 & 1 11/16×3	Yes	3/32×½ F	Eureka..........	0.35	0.39	0.28	0.34	Same as No. 1.
							Mineral Seal	0.40	0.31	0.35	0.35	Gauzes clean.
							Sperm..........	0.45	0.37	0.38	0.40	Same as No. 3.
							Lard and kerosene	0.43	0.37	0.32	0.37	Good.
8	Clanny, bonneted.	Base of bonnet......	1	1⅝ top 1 15/16 bottom ×3	Yes	5/16 R	Eureka..........	0.35	0.35	0.36	0.35	Same as No. 1.
							Mineral Seal	0.35	0.31	0.32	0.33	Top of gauze choked.
							Sperm..........	0.30	0.29	0.39	0.33	Same as No. 3.
							Lard and kerosene	0.40	0.34	0.30	0.35	Good.

TABLE 1.—(*Continued*)

9	Marsaut with Stokes shut-off.	Base of bonnet, or lower holes at top of bonnet.	2	1¾×2×3¼ 1⁹⁄₁₆×1¹¹⁄₁₆×3	Yes	³⁄₃₂×½ F	Eureka	0.32	0.31	0.31	0.31	Same as No. 1.
							Mineral Seal	0.36	0.34	0.35	0.35	Inner gauze slightly choked.
							Sperm	0.30	0.30	0.36	0.32	Same as No. 3.
							Lard and kerosene	0.43	0.24	0.32	0.33	Good.
10	Beard-Deputy	Below flame	2	1½&1¹¹⁄₁₆×3⅛ 1¼&1¾×3¾	Yes	⁵⁄₁₆ R	Eureka	0.26	0.32	0.33	0.30	Same as No. 1.
							Mineral Seal	0.33	0.31	0.35	0.33	Gauzes clean.
							Sperm	0.21	0.27	0.16	0.21	Same as No. 3.
							Lard and kerosene	0.40	0.29	0.25	0.31	Good.
10a					No		Lard and kerosene	0.41	0.31	0.34	0.35	Good.
11	Special	Below flame or through base of bonnet.	1	1⁹⁄₁₆ top 1¹¹⁄₁₆ bottom ×2½	Yes	³⁄₃₂×½ F	Eureka	0.38	0.36	0.35	0.36	Same as No. 1.
							Mineral Seal	0.47	0.46	0.49	0.47	Good.
							Sperm	0.32	0.23	0.33	0.29	Same as No. 3.
							Lard and kerosene	0.49	0.47	0.35	0.44	Good.
12	Ashworth-Hepplewhite-Gray	Below flame through top or bottom of air tubes.	1	1 top 1⅛ bottom ×2¾	Chimney	¼ R	Eureka	0.40	0.32	0.34	0.35	Same as No. 1.
							Mineral Seal	0.34	0.10	0.12	0.19	After 4 and 8 hr. glass very smoky, lamp had to be opened and cleaned. Top and one side of gauze choked.
							Sperm	0.26	0.30	0.26	0.27	Same as No. 3.
							Lard and kerosene	0.44	0.26	0.26	0.32	Fair.
13	A-H-G Improved, 6 air tubes.	Below flame through top or bottom of air tubes.	1	1 top 1³⁄₁₆ bottom ×2⅜	Chimney	⅛×½ F	Eureka	0.57	0.47	0.50	0.51	Same as No. 1.
							Mineral Seal	0.49	0.50	0.55	0.51	Gauze clean.
							Sperm	0.51	0.56	0.54	0.54	Same as No. 3.
							Lard and kerosene	0.66	0.44	0.49	0.53	Good.
14	A-H-G Improved, 1 large air tube.	Below flame through top or bottom of air tubes; top of tube opens into bonnet.	2	1⁹⁄₁₆×2¼ 1⁵⁄₁₆×2	Bonnet	⅝×³⁄₃₂ F	Eureka	0.43	0.36	0.40	0.40	Glass smoked.
							Mineral Seal	0.52	0.44	0.47	0.48	Glass smoked, gauze slightly choked.
							Sperm	0.52	0.46	0.45	0.48	Same as No. 3.
							Lard and kerosene	0.53	0.48	0.47	0.49	Good.
15	Koehler	Below flame	2	2 & 2³⁄₁₆×3¹¹⁄₁₆ 1⅝ & 1¹¹⁄₁₆×3¹³⁄₁₆	Bonnet	⁹⁄₃₂ R	Gasoline 72° Bé.	0.64	0.65	0.60	0.63	Gauzes clean.
17	Koehler	Below flame	2	2 & 2³⁄₁₆×3¹¹⁄₁₆ 1⅝ & 1¹¹⁄₁₆×3¹³⁄₁₆	Bonnet	³⁄₃₂×⅝ F	Gasoline 72° Bé.	0.66	0.64	0.66	0.65	Gauzes clean.
18	Cremer	Below flame	2	1¾ & 2×3½ 1¼ & 1⁹⁄₁₆×3½	Bonnet	¹⁄₁₆×⅝ F	Gasoline 72° Bé.	0.72	0.70	0.67	0.70	Gauzes clean.
19	Wolf	Below flame	2	1⁹⁄₁₆ & 1⅞×3⅝ 1⁵⁄₁₆ & 1⅝×3½	Bonnet	⁹⁄₃₂ R	Gasoline 72° Bé.	0.59	0.60	0.57	0.59	Gauzes clean.
20	Wolf	Below flame	2	1½ & 1¾×3⅝ 1¼ & 1⅝×3⁷⁄₁₆	Bonnet	³⁄₃₂×⅝ F	Gasoline 72° Bé.	0.64	0.59	0.61	0.61	Gauzes clean.

Lamp No. 4 is an ordinary improved Clanny unbonneted with brass oil vessel and frame.

Lamp No. 5 is constructed on the principle of a Clanny, unbonneted, but the gauze consists of perforated sheet copper instead of woven wire. It is also equipped with a Hailwood porcelain flat burner and brass oil vessel and frame.

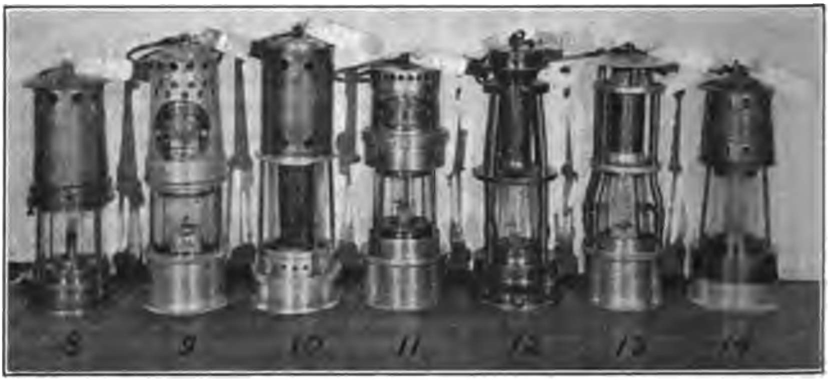

Fig. 3.—Lamps Nos. 8 to 14 Tested for Illuminating Power.

Lamp No. 6 is a bonneted Clanny very similar to the lamp known as the Evan Thomas No. 7. The air is admitted through openings in the lower part of the bonnet, and a deflecting ring is placed between these openings and the gauze, requiring the air to pass upward and over this deflector. The oil vessel and frame is brass and bonnet is of sheet iron.

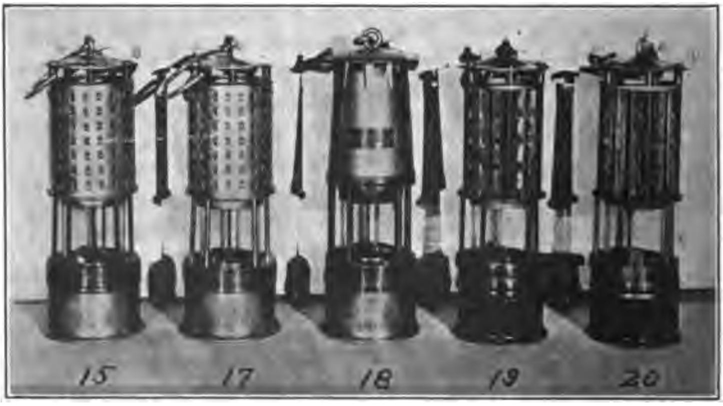

Fig. 4.—Gasoline-Burning Lamps Tested.

Lamp No. 7 is a standard Hailwood working lamp constructed on the Marsaut principle with two gauzes. Oil vessel and frame of brass, bonnet of sheet iron, and porcelain flat-wick burner.

Lamp No. 8 is a bonneted Clanny with smoke gauze.

Lamp No. 9 is a Marsaut type with Stokes shut-off ring at base of bonnet by which air is admitted at that point or, when closed, forced to enter the lower set of holes near the top of bonnet and pass down behind a deflecting ring extending from just above the inlet openings to about the middle of the bonnet. The lamp is fitted with Hailwood flat-wick porcelain burner.

Lamp No. 10 is a Beard Deputy with Beard-Mackie sight indicator. The circulation of air is on the Eloin principle, the air entering through gauze-covered openings around the base below the flame.

Lamp No. 11 is a testing lamp of English manufacture. By means of a sliding ring around the base the air may enter gauze-covered openings below the flame, or, when these are closed, it enters through openings in the lower part of the bonnet passing through the lower part of the gauze as in the ordinary Clanny lamp.

Lamp No. 12 is an ordinary Ashworth-Hepplewhite-Gray.

Lamp No. 13 is an improved Ashworth-Hepplewhite-Gray with Hailwood porcelain flat-wick burner and six hollow inlet tubes from top to bottom, each with an opening at the bottom, instead of four inlet tubes as in the ordinary form No. 12.

Lamp No. 14 is an improved Gray lamp with brass flat-wick burner. A single hollow standard of large size connects the gauze-covered air inlet in the base with the interior of the bonnet. This standard has an opening near the bottom with sliding sleeve to cover it. The gauze is of much larger diameter than those of Nos. 12 and 13.

Lamps No. 15 and 17 are respectively round and flat-wick Koehler's, burning gasoline.

Lamp No. 18 is a flat-wick Cremer, burning gasoline, differing from the Koehler and Wolf in the construction of the bonnet. The air enters through slots in the circumference of the base of the bonnet and not through perforations along the sides. It has a corrugated expansion ring between the base of bonnet and flange of the gauzes similar to the Koehler.

Lamps No. 19 and 20 are respectively standard round- and flat-wick Wolf's.

The five gasoline lamps are all fitted with Pyro igniters.

The illuminating power of some of the types tested as quoted by other authorities is given in Table 2. A wide variation in some cases will be noted. It may be proper to mention at this point that some of the results obtained were checked by testing the same lamps with a Bunsen type photometer, the results agreeing closely. The greatest difference will be noted in Nos. 10 and 12. No. 10 was an aluminum Beard-Deputy with Beard-Mackie sight indicator. To check results on this, No. 10*a* was tested with one oil. No. 10*a* lamp was identical except that it was made of brass, the indicator was omitted, and it did not have a bonnet. The results on No. 12, A-H-G lamp, were checked with a duplicate lamp, the results agreeing within 5 per cent. The good lighting power of No. 13 over No. 12 is undoubtedly due to the better air supply of No. 13. As noted in Table 1, this lamp has six air tubes each with an air hole at the base, whereas No. 12 has four tubes with air inlets near the base in two only. A considerable difference will also be noticed between the observed candlepower of the Wolf type lamps and the generally accepted candlepower with a value of 1.

Table 2.—Illuminating Power of Certain Lamps According to Different Authorities

(Expressed in Candlepower)

Type	Authority								
	Marsaut	Beard	Coal & Metal Miner's Pocket Book	Mines Accident Commission	Crossley	J. A. Longden	Prussian Fire Damp Com.	Lupton, Parr & Perkins	Average
Davy (common)	0.18	0.15	0.16	0.126–0.15	0.125	0.10	.....	.....	0.141
Clanny (various types)	.	0.25–0.50	0.20	....	...	..	....	....	0.317
Clanny, unbonneted	0.47	.....	....	0.50	0.23	0.21	0.62	....	0.416
Clanny, bonneted	..	.....	....	0.31–0.64	...	0.25	....	....	0.40
Evan Thomas No. 7	..	0.40	0.45	0.46	...	..	....	....	0.437
Marsaut (2 gauze)	..	0.60	0.55	....	...	..	....	....	0.575
Ashworth-Hepplewhite-Gray	..	0.65–0.79	....	....	0.65	..	....	1.00	0.77
Beard, Eloin-Marsaut (2 gauze)	..	0.75	....	. .	...	..	....	....	0.75
Wolf	..	0.80–1.00	0.90	....	...	..	0.66	0.40	0.75

Conditions of Test

After being cleaned, filled, and lighted, the flame of each lamp was adjusted to a normal height suitable for working conditions, as near as it was possible to judge. This was ordinarily done by turning up the flame until it would begin to smoke and then slowly lowering it until it stopped smoking. The lamps were then allowed to stand 20 to 30 min.

or until all parts were thoroughly warmed up, before the first readings were taken. The flames were again adjusted just before the readings were made. They were also maintained at approximately a working height throughout the time of 8 hr. and carefully adjusted at each observation.

Illuminants

The oils used were Eureka Safety Lamp oil, Mineral Sea oil, pure sperm oil, pure lard oil, a mixture of lard oil and kerosene, and gasoline. The first two are mixed oils sold on the market under the trade names given. Eureka Safety Lamp oil is apparently a mixture of lard or cotton seed, possibly both, and a petroleum oil. Mineral Seal is a mixture of fish or sperm oil and petroleum oil with possibly other ingredients. The mixture of lard oil and kerosene was prepared for the purpose of this test and was composed of 50 per cent. lard oil and 50 per cent. kerosene by volume. Pure lard oil was tried in all the lamps, but even after they were warmed up such constant attention was required to keep them burning properly that any results obtained would have been of no practical value.

The specific gravities of the oils and the corresponding gravity on the Baumé scale are given in Table 3.

TABLE 3.—*Specific Gravity of Safety-Lamp Oils at 18° C.*

Oil	Specific Gravity, Water = 1.00	Degrees, Baumé
Eureka Safety Lamp	0.863	32
Mineral Seal	0.818	41
Sperm	0.900	25
Lard	0.913	23
Lard 50 per cent.; kerosene 50 per cent.	0.860	32
Gasoline	0.695	72

Conclusions

The average candlepower for the different oils indicates clearly the advantage of a mixed oil or the addition of petroleum oil. The average illuminating powers of lamps Nos. 1 to 14 for the different oils, excepting gasoline, are given in Table 4.

TABLE 4.—*Average Illuminating Power of Different Oils*

Oil	Average Candlepower	Specific Gravity of Oil
50 per cent. lard, 50 per cent. kerosene	0.333	0.860
Eureka Safety Lamp	0.318	0.863
Mineral Seal	0.316	0.818
Sperm	0.296	0.900

There is apparently no definite variation of candlepower with the specific gravity of the oil, for although sperm oil is the heaviest and has the lowest average illuminating power, Mineral Seal, the lightest, is intermediate in illuminating power between sperm and the other two heavier oils.

In addition to better lighting, the mixed oils burned more freely and required less attention to the wick and flame, but showed a slightly greater inclination to smoke than the pure sperm oil. The better illuminating power of the mixed oils is especially noticeable in Lamps Nos. 4, 6, 10, and 11. The effect of a bonnet in decreasing illuminating power is indicated to some extent in Lamps Nos. 10 and 10*a*, identical except for a bonnet. With the mixed lard and kerosene a difference of 0.04 candlepower is noted.

The effect of admitting the air below the flame or even at the base of the bonnet, instead of higher up, is shown conclusively in lamps Nos. 9 and 11, constructed so that air could be admitted in different ways. In No. 9, the results given in Table 1 were obtained with air entering at base of bonnet; the same lamp with air entering near the top of the bonnet and passing downward between bonnet and deflecting ring gave an average candlepower of 0.24 with Eureka oil, 0.07 less than the value given in Table 1. With No. 11 the results are even more pronounced. With the inlet at the bottom closed, and air entering at base of bonnet and gauze, the average candlepower with Eureka oil was 0.23, while with air entering below the flame it was 0.36, a difference of 0.13 or 36 per cent.

In testing the lamps with flat wicks, the candlepower was in all cases taken in a line normal to the width of the flame. The decrease in lighting edgewise to the flame was much less than anticipated. For the oil lamps the candlepower edgewise varied from 5 to 20 per cent. less with an average of 10.5 per cent. For the gasoline lamps the illumination edgewise was 4 to 7 per cent. less.

With the exception of gasoline, the decrease of candlepower during the 8 hr. seems to vary inversely as the illuminating power of the oil. Sperm oil gave an average candlepower for 14 lamps when freshly lit of 0.301 and after 8 hr. of 0.304, a gain of 1 per cent.; Eureka oil gave at the beginning an average of 0.323 cp. and after 8 hr. 0.308 cp., a decrease of 4.3 per cent.; Mineral Seal at beginning an average of 0.340 cp. and after 8 hr. 0.314 cp., a decrease of 7 per cent.; a mixture of 50 per cent. lard and 50 per cent. kerosene at the beginning gave an average of 0.403 cp. and after 8 hr. 0.296 cp., a decrease of 26.5 per cent. To determine whether the decrease in candlepower was due to the lighter part of the oil burning off first, leaving a heavier residue, the specific gravity of the oil remaining in the lamps after burning the mixture of kerosene and lard oil was determined. This was found to be 0.865 or 0.005 greater than the original oil. As this is only 0.58 per cent. gain, it is within a possible experimental error, and it would be hardly safe to assume from it that there was a separation of the oils. It would be more reasonable to conclude that the decrease in illumination was due to the more rapid deposit of soot on the gauze and a film of unburned carbon on the glass when burning oils containing petroleum products, than when burning pure sperm oil.

With the gasoline lamps, the illuminating power decreased in 8 hr. from an average of 0.65 to 0.62 cp. or 4.6 per cent.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Iron Pyrites Deposits in Southeastern Ontario, Canada

BY P. E. HOPKINS,* B. A., B. S., TORONTO, ONT.

(Arizona Meeting, September, 1916)

Introduction and History

In speaking of the economic geology of southeastern Ontario, W. G. Miller and C. W. Knight[1] say that "there occurs in southeastern Ontario a variety of minerals and rocks of economic value, probably as great as in any district of like size on the North American continent. Some of these deposits, including marble and trap, are inexhaustible. Others, including talc and iron pyrites, have proved to be of considerable economic importance. From time to time, during the last 50 years, the following minerals and rocks have been mined or quarried with varying success: gold, iron pyrites, zinc blende, copper pyrites, galena, mispickel, magnetite, hematite, talc, actinolite, mica, marble, ophicalcite, feldspar, fluorite, apatite, corundum, graphite and sodalite. All of the economic materials, with the exception of fluorite, appear to be of pre-Cambrian age. The fluorite veins penetrate the Ordovician, Black River, limestone."

Accompanying that report was an article by the writer on the Queensboro Pyrite Area which includes one of the two working pyrite properties in southeastern Ontario.

In the present paper will be given a brief description[2] of all the known pyrite deposits in the area which may at some time possess an economic value, with fuller descriptions of the two working mines—The Canadian Sulphur Ore Co.'s mine near Queensboro, and the Nichols Chemical Co.'s property at Sulphide.

The earliest mining of iron pyrites in Ontario was done in 1868 on the Billings property near Brockville. The mines were closed down in 1879 under the assumption that they were exhausted. Many other

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[1] The Pre-Cambrian Geology of Southeastern Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 22, part 2 (1913).

[2] The information regarding the various pyrite prospects in southeastern Ontario is summarized from E. L. Fraleck's comprehensive report on Iron Pyrites in Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 16, part I, pp. 149–201 (1907).

pyrite deposits have been worked for gold, iron or copper at some time. The steady pyrite industry of the Province began in 1900 when ore from the Bannockburn mine was produced. Mines in Hastings County have been steady producers since that time. An acid-making plant has been in operation at Sulphide since 1907 by the Nichols Chemical Co. for the treatment of its ore at Sulphide. The company also buys the ore mined from other properties in the neighborhood. Another plant for treating custom ore is operated by the Grasseli Chemical Co. at Hamilton. These two plants treat the bulk of the eastern Ontario production, the remainder being shipped to the United States.

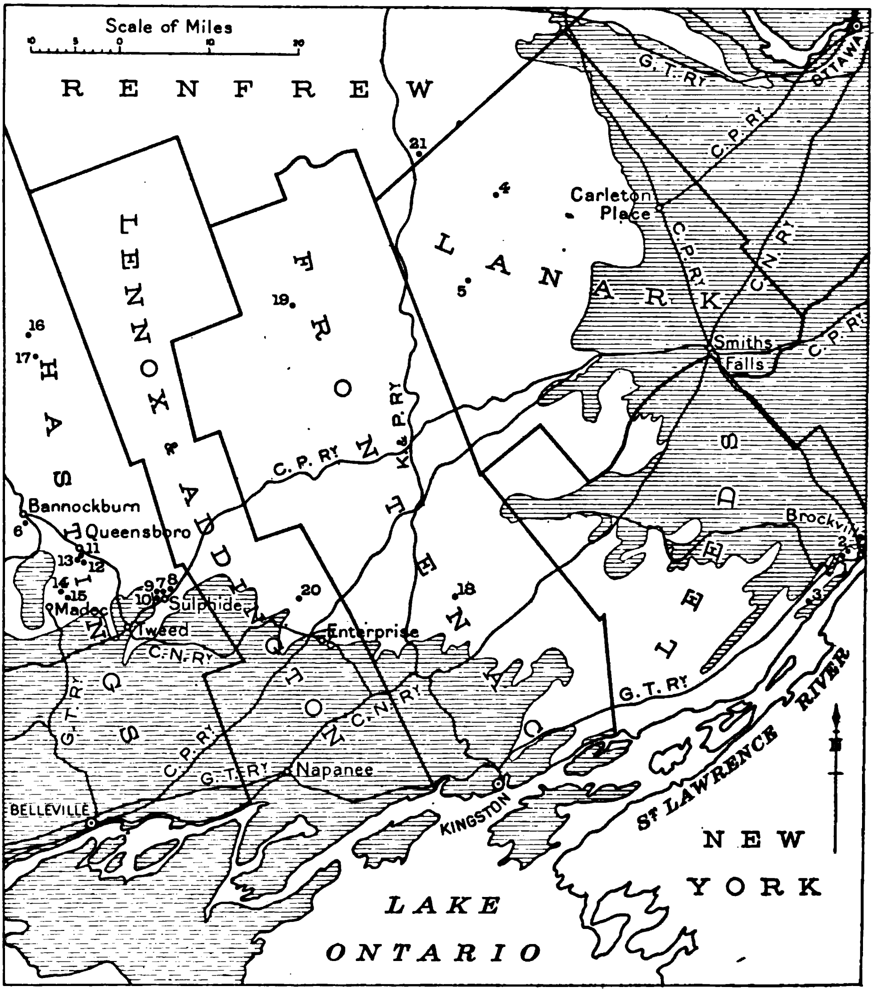

FIG. 1.—SKETCH MAP SHOWING PRE-CAMBRIAN AND PALEOZOIC (HATCHED) AREAS IN SOUTHEASTERN ONTARIO. THE NUMBERS INDICATE THE LOCATIONS OF PROPERTIES DESCRIBED.

Recently a large percentage of the production has been coming from the Vermilion Lake deposits[3] in northwestern Ontario, the ore being shipped to United States ports on the great lakes. Another property, the Goudreau Lake deposits,[4] has been recently developed and expects to

TABLE 1.—*List Showing the Locations of Pyrite Deposits in Southeastern Ontario*[5]

Brockville Section.
- † 1. Brockville Chemical Co. (Billings property); lot 19, con. 2, Elizabethtown township.
- † 2. Sloan prospect; lot 18, con. 2, Elizabethtown township.
- 3. Shipman prospect; about 6 miles west of the Billings (No. 1).

Lanark County.
- † 4. McIlwraith mine; lot 5, con. 4, Darling township.
- 5. Ladore prospect; lot 19; con. 7, Dalhousie township.

Hastings County.
- ‡† 6. Bannockburn (or Jarman) mine; lot 25, con. 6, Madoc township.
- * 7. Hungerford mine (Nichols Chemical Co.); lot 23, con. 12, Hungerford township.
- † 8. Canada mine (formerly Oliver Prospect); lot 26, con. 12, Hungerford township.
- 9. Hungerford Western Extension; parts of lots 21 and 22, con. 12, Hungerford township.
- † 10. Ontario Sulphur Mines, Ltd.; northwest quarter of east half of lot 21, con 12, Hungerford township.
- † 11. Queensboro mine; lot 11, con. 11, Madoc township.
- * 12. Canadian Sulphur Ore Co. (formerly Wellington prospect) N. ½ lot 9, con 10, Madoc township.
- † 13. Davis or Palmer prospect; lot 10, con. 10, Madoc township.
- 14. Farrell prospect; 2 miles northeast of Madoc village.
- ‡ 15. McKenty prospect; 2 miles east of Madoc village.
- 16. Little Salmon Lake deposit; lot 23, con. 7, Cashel township.
- 17. Gunter property; lot 23, con. 4, Cashel township.

Other Eastern Ontario Prospects.
- 18. Snooks prospect; lot 7, con. 14, Loughborough township, Frontenac Co.
- 19. Stalker prospect; lot 42, con. 6, Clarendon township, Frontenac Co.
- 20. Foley prospect; 5½ miles north of Enterprise Sta., Lennox, Addington Co.
- 21. Caldwell prospect; lot 1 con. 1, Blithfield township, Renfrew Co.

* Mines now working (April, 1916) and shipping pyrites.

† Properties which have shipped pyrites.

‡ Properties which have shipped hematite or limonite.

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[3] E. S. Moore: Vermilion Lake Pyrite Deposits, *Annual Report, Ontario Bureau of Mines*, vol. 20, part I, pp. 199–209 (1911).

T. F. Sutherland: Northern Pyrites Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 94–95 (1915).

[4] A. L. Parsons: Goudreau Pyrite Claims, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 211 (1915).

T. F. Sutherland: Madoc Mining Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 107 (1915).

[5] The number of each property refers to the corresponding number showing its position on the accompanying map.

commence at once supplying large tonnages. The Helen mine,[6] operated by the Algoma Steel Corporation, produces some pyrite which is treated in its plant at Sault Ste. Marie.

The iron pyrites resources of Ontario are of considerable extent and value, in the last 15 years 538,755 tons, worth $1,438,122, having been produced, the greater part coming from southeastern Ontario. During the coming years there will undoubtedly be a steady increase in production. The war has had a stimulating effect on the demand of the United States for pyrite from Ontario.

Brockville Section

The Brockville Chemical Co., No. 1,[7] began mining for pyrite on the Billings property in 1868. The ore occurred in a series of lenses conformable to the lamination of a highly foliated pink granite gneiss. The lenses, which consist of pyrite and calcite in parallel lines, strike northeast and dip to the southeast. The richer shoots of ore were gouged out and no timbering was done. The main pit was sunk 250 ft. The ore was used for making acids in Brockville, the sulphuric and mixed acids being used at the fertilizer and dynamite works in and near Brockville. Operations of all kinds ceased in 1880. The evidence of the men who worked in the old pits is to the effect that they were never completely exhausted.

Sloan Prospect, No. 2.—A band of gossan strikes in a northeast direction across the property and dips to the southeast. The 20-ft. inclined shaft passes through 6 or 8 ft. of gossan. There is a width of 3 ft. of solid pyrites on the foot wall, the remainder of the shaft being in alternating bands of pyrite and crystallized calcite in equal amounts. Eighty tons of ore, running 40 per cent. sulphur, were shipped to Buffalo and Capelton.

The Buffalo-Brockville Mining Co. shipped a small tonnage from this lot during 1911 and 1912.

Shipman Prospect, No. 3.—The pyrite, which is much intermixed with pyrrhotite and country rock (gneiss), has been mined from an irregular pit 40 ft. long and 30 ft. wide.

Lanark County

McIlwraith Mine, No. 4.—The deposit, which is covered by 14 ft. of gossan, strikes north of east along a contact between diorite on the south and crystalline limestone on the north, and dips 60° to the south. It was first opened for gold. In 1899 and 1900 the shaft was deepened to 75 ft. and a 150-ft. tunnel run along the strike of the deposit, disclosing a length of over 90 ft. of clean high-grade pyrite inclosing lenses of quartz. A 12-ft. crosscut to the south did not pierce the width of the deposit. Three carloads of ore were shipped. Samples from the dump and tunnel, by E. L. Fraleck, gave 38.86 and 42.60 per cent. of sulphur respectively.

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[6] A. L. Parsons: Helen Mine, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 202–205 (1915).

[7] The numbers following mention of the pyrite properties refer to corresponding numbers showing their positions on the accompanying map, and in Table 1.

Ladore Prospect, No. 5.—A heavy fahlband strikes north of east along the contact of a coarse amphibolite and a fine-grained gray granite. The trenches and shallow pits expose a gossan in the form of bog iron ore, but pyrite in quantity was not located. The fahlband continues into the adjoining lot to the east along a contact of crystalline limestone and granite.

Bannockburn Mine, No. 6.—In 1898, the property was opened as an iron mine, 11 car loads of limonite, running about 38 per cent. in iron and low in sulphur, having been shipped. This ore was merely a gossan 8 to 15 ft. deep which capped iron pyrites deposits. The pyrites occurred as two lenses at right angles to each other, but conforming in strike and dip with the inclosing rock, a chloritic schist. Limestone covers the apex of the fold of the lenses. The south lens, which is 160 ft. long and 8 to 15 ft. wide, was mined to a depth of 275 ft. During the 6 years of operation about 580 tons of pyrite per month were shipped, all of which went to the General Chemical Co. at Buffalo. The ore did not fall off either in grade or quantity with depth, but, owing to the hazard of open-pit mining, operations were abandoned in August, 1906.

Hungerford Mine, No. 7.—This property was opened 40 years ago as a gold property, and a smelter was erected to extract gold from the barren pyrite. The Nichols Chemical Co.[8] re-opened the mine in June, 1903. Owing to some difficulty about the title, the mine was closed down in August, 1904, but operations were resumed in August, 1905, and have since been continuous. Since 1907 acid works have been in operation for the treatment of company ores, and other ores in the vicinity.

Passing through this property, and extending beyond, is a large fahlband striking 25° north of east and traceable for 2 miles. Level farm land to the south is underlain by garnetiferous crystalline schist cut by massive diorite, into which, 500 yd. north of the deposits, has been intruded a pink hornblende granite that rises above the country in a series of rugged hills, locally called the Bald Mountains. The granite has protected the deposits from denudation. The deposits are strung along the contact of the diorite and the schist, the strike of lenses, contact, fahlband, and schist being identical.

The pyrite occurs in three parallel deposits striking with the schist and dipping 60° to the south. The middle one, which does not outcrop on the surface, lies 85 ft. from the south vein and 45 ft. from the north deposit. The north deposit, upon which most of the work has been done, varies in width from 6 to 22 ft. It has been exploited to a length of 620 ft. and to a depth of 575 ft., and the ore still continues. The length as indicated on the surface is about 500 ft. There are now two shafts on the property and about 3,500 ft. of drifting has been done on the orebodies on the six levels.

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[8] W. H. Nichols, President, 25 Broad St., New York.

The ore is coarsely granular and makes a large percentage of fines. The main impurity is calcite, although there is also some quartz present. A small quantity of pyrrhotite occasionally occurs, mainly in the north lode next the foot wall. The average percentage of run of mine ore is about 35 per cent., the fines being much higher.

The acid works have been successfully operated since their completion in July, 1907, and machinery has been installed at various times to increase the capacity and to make new acids. At present sulphuric, hydrochloric, nitric and mixed acids are made by the contact process and shipped in the company's tank cars to various parts of Ontario and Quebec.

Electric power supplied by the Seymour Power and Electric Co. is used throughout the mine and acid works.

The Canada Mine, No. 8, which was formerly the Oliver prospect, adjoins the mine operated by the Nichols Copper Co. on the east, and is located on the same fahlband. The lode strikes east and west and dips 50° to the south. During part of 1907, the Canadian Pyrites Co. sank an inclined shaft on the deposit to a depth of 110 ft. and did some drifting on the 85-ft. level, together with some diamond drilling. The deposit varies from 4 to 7 ft. in width. The ore on the dump is pyrite with a little pyrite and pyrrhotite, which will grade upward of 40 per cent. in sulphur.

The Hungerford Western Extension, No. 9, was fairly well prospected in 1906 by means of surface trenches at regular intervals along the strike of the fahlband. The western lens had been exploited by surface trenches to a length of 500 ft., exhibiting, near the line between the lots, a width varying from 16 to 18 ft. of ore, which will grade from 42 to 44 per cent. sulphur. The only impurity consists of small included lenses of calcite.

The eastern lenses are presumably continuations of the Hungerford mine orebodies.

A gossan 40 ft. wide occurs on the south end of the property, but not enough work has been done to determine the extent of the deposit.

The Ontario Sulphur Mines, Limited,[9] *No.* 10, commenced work in

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[9] Formerly the Craig property.

March, 1908, and continued until the end of 1911, save for 2 months in the summer of 1910. The pyrite deposit on which work has been done is located about ½ mile west of the Hungerford mine. It appears to be a lens pitching toward the southeast. A shaft has been sunk 300 ft., with 225 ft. of drifting on the 100-ft. level and 250-ft. on the 200-ft. level. According to A. W. G. Wilson,[10] "The total shipments from the property up to the first of May, 1911, have been 4,821 long tons of ore averaging 36½ per cent. sulphur." In one place the deposit is 30 ft. wide.

The Sulphide Chemical Co. operated the property from the spring of 1914 until the following November, during which time the mine was dewatered and considerable ore was raised and shipped.[11] No work has been done since.

The Queensboro Mine (*Blakely*), *No.* 11, up to the autumn of 1906 shipped 65 carloads of pyrites running about 45 per cent. sulphur. Mine operations ceased in 1908. The pyrite occurs as a series of lenses up to 15 and 20 ft. wide along the contact of a garnetiferous schist (Grenville in age) and an intrusive pink felsite (post-Hastings in age). The ore is dense, the only impurity being thin veinlets of quartz. Cutting a pyrite lens is a small quartz vein containing copper pyrites and argentiferous jamesonite. In another place some zinc blende is interbanded with the pyrite. The main shaft is 135 ft. deep with about 175 ft. of drifting on the 50- and 85-ft. levels.

The Canadian Sulphur Ore Co.'s Pyrites Mine,[12] *No.* 12, was discovered in 1906 by Stephen Wellington while prospecting for iron. Under the gossan, merchantable iron pyrites was discovered, from which a car load of iron pyrites was shipped in 1908. Later, the Canadian Pyrites Syndicate bought the property, installed a small plant and shipped a few hundred tons of pyrite. In the spring of 1910 the property was handed over to the present company, which began shipping ore 3 months later, and has continued to the present. The mine is equipped to produce 100 tons of iron pyrites per day, yielding 40 per cent. of sulphur. Since Dec. 11, 1912, the mine has been run by electricity supplied by the Seymour Power Co. A branch line 2½ miles in length from the Bay of Quinte Railway near Queensboro to the mine was completed in 1913. The ore is shipped to the Nichols Chemical Co.'s acid plant at Sulphide, 11 miles southeast, and to the chemical companies at Hamilton and Detroit.

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[10] A. W. G. Wilson: Pyrites in Canada, *Publication No.* 167, *Canada Mines Branch*, p. 66 (1912).

[11] A. W. G. Wilson: *Annual Report, Ontario Bureau of Mines*, vol. 23, part I, p. 174 (1914).

[12] Alex. Longwell, President, 410 Crown Office Building, Toronto. For a fuller description see P. E. Hopkins: The Queensboro Iron Pyrites Deposits, *Annual Report, Ontario Bureau of Mines*, vol. 22, part II, pp. 89–104 (1913).

The pyrite is mined by underground and open-pit methods. The development work consists of three shafts and two open cuts, with some diamond-drill borings. Nos. 1 and 2 shafts, which are 75 and 100 ft. deep respectively, have been abandoned for some time. The work in late years has been confined to shaft No. 3 and the two open pits. The vertical shaft, No. 3, is 250 ft. deep with about 800 ft. of drifting on the 60-, 120-, and 200-ft. levels. The pyrite deposits are marked by gossan outcrops from 2 to 30 ft. in depth. Beneath are the pyrite deposits, which occur as lenses in contact with rusty schist to the south and white quartzite to the north (both Grenville in age) near an irregular post-Hastings intrusion of gray felsite. The strike of the deposits is slightly north of east, while the dip is almost vertical, inclining slightly to the south. Lenses vary in width up to 25 ft., but horses of country rock are frequently inclosed in the pyrites.

The ore is high grade, very little cobbing, if any, having to be done. Ores have been shipped running 40 to 48 per cent. sulphur.

The deposits are free from impurities such as arsenic, zinc, lead, copper and calcium. The pyrite burns satisfactorily, and is in good demand by sulphuric acid makers.

The Davis or Palmer Deposit, No. 13, is in the Grenville limestone. On the surface the pyrite is 2 ft. in width; 9 ft. down there is said to be a deposit 15 ft. wide. A few carloads of pyrites were shipped from a pit sunk on the property.

The Farrell Deposit, No. 14, lies in and conforms with the schist which strikes northwest. Test pits for a distance of 200 ft. show either gossan or pyrite. A shaft has been sunk to a depth of about 25 ft. A sample collected by E. L. Fraleck, representing an average of 75 per cent. of the dump (which consists of about 40 tons), yielded 40.64 per cent. of sulphur. The deposit maintains a uniform width of 5 ft., the only impurity being crystalline limestone.

The McKenty Prospect, No. 15, shipped hematite 40 years ago. A pit at one time 60 ft. deep has caved in. An examination of the cull dump reveals the fact that all large lumps of apparent hematite have, when broken, a core of pyrites. In E. L. Fraleck's opinion, this is one of many instances throughout eastern Ontario where hematite constitutes the gossan capping of a sulphide orebody.

The Little Salmon Lake Deposit, No. 16, occurs in a chlorite schist which strikes east and west, the main rock in the area being a white crystallized limestone, probably of Grenville age. A trench, 40 ft. long, uncovered pyrite 15 ft. in width. An average of 75 per cent. of the pyritiferous material yielded 38.83 per cent. of sulphur.

On the *Gunter Property, No.* 17, a shaft, 20 ft. deep, has been sunk on a deposit consisting of alternating bands of quartz and pyrite 5 ft.

wide. A sample representing two-thirds of the dump yielded 39.50 per cent. of sulphur.

Snooks Prospect, No. 17.—A fahlband strikes northeast through a coarse, impure crystalline limestone, and can be traced across the adjoining lot 6 to Desert Lake. On the road allowance, 7 ft. of massive pyrite and 25 ft. of pyrite mixed with crystalline limestone were uncovered in obtaining material for the road.

On the *Stalker Prospect, No.* 19, is a well-defined fahlband, containing some hematite, and striking east and west. A small test pit has been sunk on a lens of pyrite which shows at that point a width of 6 ft.

The Foley Deposit, No. 20, occurs in an outlier of crystalline limestone surrounded on all sides at short distances by granite. The irregular deposit consists of small masses of pyrite and pyrrhotite in about equal proportions. The work consists of a pit, 80 ft. long, 40 ft. wide and 10 to 15 ft. deep, sunk on pyrite and pyrrhotite in about equal proportions intermixed with pyroxene, calcite, mica and molybdenite.

The Caldwell Prospect, No. 21, was opened in the fall of 1915 by Thomas B. Caldwell of Lanark. About 500 tons of ore have been mined, but the sulphur contents are not known.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Method of Mining Talc

BY F. R. HEWITT, ASHEVILLE, N. C.

(Arizona Meeting, September, 1916)

THE methods of mining talc are simple, and in western North Carolina are almost entirely by open cut and quarry. The larger part of the talc of this section lies in various-sized "veins" inclosed in quartzitic walls, the majority of which have by folding been thrown into a perpendicular position, or nearly so, and the exposed edges of which have been covered by débris from erosion of the mountain above. This covering of loose rocks and earth is from 5 to 25 ft. thick, and in most cases has to be removed because it is difficult to support. Sometimes the covering is heavy enough to "catch" and timber successfully, and in this case the talc can be worked out by the usual method of following the "vein," using stulls for holding up the walls until the vein is worked out, and then allowing the cut or drift to fall in.

In some cases it proves more convenient to sink shafts and run drifts, following the talc "veins" until exhausted. The talc deposits of this section are badly broken and faulted both laterally and perpendicularly, and in some cases are found many feet below the level of drainage, necessitating pumping at considerable expense.

My own experience in mining talc has been that the chief point is to get a reliable "vein," and that then the mining is not as difficult, or any more so at least, than that of any common ore. If the "veins" are pure talc it is simple; if the talc is admixed with foreign matter, such as tremolite in excess, or stains of iron or manganese, as is often the case, one had better abandon the deposit.

Methods of Manufacture

Talc, being a soft mineral, is not difficult to reduce to the condition desired by the trade. Powdered talc is admitted under low duty from France, Italy and Austria, and some is being imported from Asiatic points. Talc is usually ground and bolted in any simple reduction mill. That mined in this section is hand-sorted and the quality that will make good powdered talc is ground for use in the manufacture of talcum powder, cosmetics, etc., while all that is hard enough and has sufficient strength is used for crayons and blanks for gas burners and electrical work, being sawed into the shapes desired. The rest that is clean and pure is ground in mills, of which many different kinds are used. Some people like the mill stone, while others prefer the many kinds of high-speed beater mills for grinding.

The manufacturing process can be summed up as follows:

More or less hand-sorting may be necessary, and in grinding the talc must be reduced sufficiently to pass through a 170- to 200-mesh sieve, silk cloth reels being commonly used.

Among the objectionable impurities which it is almost impossible to remove are lumps of tremolite and pyrite. They destroy the value of the talc by changing the color and making a coarse and hard gritty product which is not saleable.

Uses

Most of the talc sold is used in the paper, rubber and paint trade, while the lower grades go largely into roofing. About one-quarter of the production goes into talcum powder. Not a little is employed as a body or carrier of medicinal chemicals used in tablet form.

Much of the talc or soapstone is cut with small saws into crayons of various sizes for use in the iron-working trades and on blackboards. A growing business is the manufacture of gas burners and small blocks for electrical work, which, after shaping and burning, are known as lava goods.

The two last, crayon and burner manufacture, are the most important uses of talc, but the greatest difficulty is to find mineral of the proper quality, as it has to be solid, firm, and free from grit and other foreign matter.

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An Explanation of the Flotation Process

BY ARTHUR F. TAGGART[*] AND FREDERICK E. BEACH,[†] NEW HAVEN, CONN.

(Arizona Meeting, September, 1916)

INTRODUCTION

THE flotation process for the concentration of ores is a method by means of which one or more of the minerals in the ore (usually the valuable ones) are picked up by means of a liquid film and floated at the surface of a mass of fluid pulp. Here they are separated from the other minerals, which remain immersed in the body of the pulp. In general, the minerals which are floated are sulphides of metallic luster, but some other minerals of metallic luster such as graphite and some sulphides with adamantine luster, such as sphalerite and cinnabar, are amenable to treatment by the process.

The importance of flotation lies in the fact that it is primarily a "slimes process" by means of which the particles of valuable mineral, too fine for efficient gravity concentration, are saved with a high percentage of recovery. Recoveries in the mills treating low-grade copper sulphide ores have been advanced 10 to 20 per cent. by the installation of the process and similar increased savings have been accomplished by the same means in mills treating sulphide ores of zinc and lead.

When finely ground ore containing sulphides mixed with a siliceous or earthy gangue is brought gently onto the surface of a body of water, in a direction forming an acute angle with the surface of the water, a considerable portion of the sulphide constituent of the ore floats on the surface of the liquid, while the gangue sinks. This is the so-called film flotation, exemplified by the Wood and Macquisten processes.

When gas bubbles are introduced into a fluid pulp composed of finely ground ore and water, to which has been added (1) a small amount of certain oils, or (2) a small amount of certain acids or acid salts, or (3) a small amount of certain alkalis or alkaline salts, or (4) a small amount of a mixture of oil with acid or alkali, the sulphide particles in the ore are brought to the surface on the gas bubbles. These collect in a froth heavily laden with sulphide particles. The gangue particles are not brought up by the bubbles, but remain in the mass of the pulp. This is the so-called froth flotation. Its variations, acid-froth, agitation-froth, and pneumatic-froth processes are discussed in detail later.

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[†] Assistant Professor of Physics, Sheffield Scientific School, Yale University.

The points to be explained in the operation of these processes are: (1) The flotation of solid particles in a liquid the specific gravity of which is less than that of the solid; (2) the preferential flotation of the sulphide portion of the mass; (3) the functions of the reagents used.

THEORY

The theory here presented in explanation of the points listed in the preceding paragraph appeals to the following physical phenomena: Surface tension, adsorption, adhesion and viscosity. The first three of these are closely related.

Surface Tension

Every liquid surface in contact with a gas or its vapor, behaves as if it were under tension. The value of this contractile force per unit width can be measured. Its value for water, 74 dynes per centimeter, is higher than for any other well-known liquid. (Liquid metals and fused salts are of course excepted.) This fictive tension is a convenient conception for many discussions and may be explained in terms of the intermolecular attractions of the substances forming the boundary. Another and very useful way of considering the phenomenon is to regard each unit of surface as having associated with it an amount of potential energy which is numerically equal to the surface tension. This relation is established as follows:

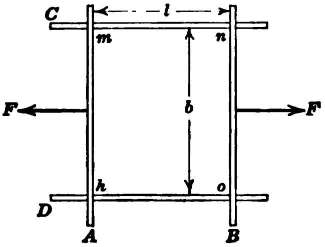

FIG. 1.

If two wires, *A* and *B* (Fig. 1), are so placed as to slide on two fixed wires, *C* and *D*, and if the wires, *A* and *B*, having been in contact, are separated a distance *l* against the pull of one of the surfaces only of a film, *mnoh*, by applying the force *F*, the work per unit area will be

$$\frac{W}{A} = \frac{Fl}{lb} = \frac{F}{b} = T$$

which is obviously equal, numerically, to the force per unit width or the surface tension, *T*, of the one surface considered. Applying this conception of surface energy to different cases of contact, we can develop some statements of the relations of forces which are important in explaining many of the phenomena observed in flotation.

Consider first that two blocks of liquid, which have become rigid without any change in their other properties, are drawn together by the mutual action of their forces of molecular attraction until they perfectly unite over an area A. They would do an amount of work that we will represent by the letter W. Now consider that the same two bodies when brought near together, but not into contact, become liquid and unite over the area A. The work done is again W, but in this case it can be considered as due to the shrinkage of the surface by an amount $2A$, whence $2AT = W = A(T + T)$, where T is the surface tension of the liquid. If two different liquids whose surface tensions are T_1 and T_2 respectively were brought together, the work due to the molecular attractions would be $(T_1 + T_2)A = W$, provided there was complete union, but if there is not complete union, there will be an interfacial tension T_{12}, and the energy equation becomes

$$(T_1 + T_2)A - T_{12}A = W$$

or

$$T_{12} = T_1 + T_2 - \frac{W}{A}$$

i.e., the interfacial tension T_{12} is the excess of the sum of the two tensions over the work which is done by them in allowing a unit area of the two liquids to come into contact. If a liquid is brought into contact with a solid, the energy equation is $T_{LS} = T_L - W_1$ (gas or vapor effects being excluded),

where T_L = the surface tension of the liquid,
W_1 = the work done in bringing a unit area of the liquid and solid into contact,
and T_{LS} = the interfacial tension.

If, therefore, $W_1 = T_L$, the solid has the same attraction for the molecules of the liquid as the molecules of the liquid have for each other and there will be no interfacial tension. If $T_L > W_1$ the interfacial tension T_{LS} will be positive; if $T_L < W_1$ there will be negative interfacial tension or a surface pressure. In the latter case the liquid will tend to spread over the solid.

Angle of Contact

When, as is the common case in the flotation process, there are three substances in contact, a system of forces as shown in Fig. 2 is brought into play. If O does not move indefinitely to the right or to the left, equilibrium will be attained when

$$T_{SL} = T_{GS} + T_{GL} \cos \theta$$

or

$$\cos \theta = \frac{T_{SL} - T_{GS}}{T_{GL}}$$

where T_{GS}, T_{GL}, and T_{SL} are the interfacial tensions or pressures at the gas-solid, gas-liquid, and solid-liquid contacts respectively. From this equation is deduced the important conclusion that as T_{SL} increases with respect to T_{GS}, the angle of contact θ becomes smaller (the gas and liquid being the same), or, in other words, the angle of contact is a measure of the tendency of one fluid to replace another on the surface of a solid. We have examined the angles of contact of the water-air, oil-air, and oil-water surfaces against a number of the common minerals. We have found, in general, that the air-water contact angle is less for gangue minerals than for sulphide minerals; that the air-oil contact angle is less for sulphides than for gangues and less for any given sulphide than the air-water contact angle; and that the water-oil contact with solids takes the form shown in Fig. 3.

We found further that the invariable effect of oiling a solid surface is to reduce the air-water contact angle. This latter phenomenon is undoubtedly aided by the reduction of the surface tension of the water due to contamination by the oil. This is further discussed later.

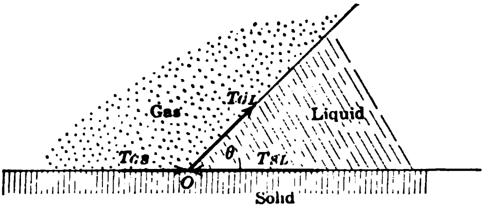

FIG. 2.

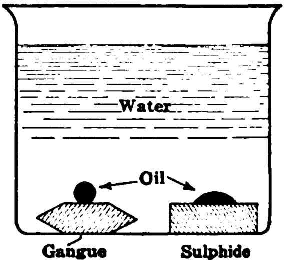

FIG. 3.

The conclusions forced by observation of the above phenomena are:

1. That water has a smaller tendency to displace air on the surface of sulphide minerals than on the surface of gangue minerals.

2. That the tendency of oil to displace air is greater at the surface of sulphide minerals than at the surface of gangue minerals.

3. That oil tends to displace water on the surface of sulphides and that water tends to displace oil at the surface of gangue minerals.

4. That water displaces air more readily on an oiled solid surface than on a clean surface of the same solid.

5. That these tendencies toward displacement are due to the interfacial tensions or pressures existing between the various substances, and that the resulting action of these interfacial forces is a manifestation of the tendency toward reduction of the total potential energy of the system. Wherever an increase in the solid-fluid interface will decrease the potential energy, such a change will occur.

These conclusions suggested the following confirmatory experiment.

A ring, 6.17 cm. outside diameter and specific gravity of 1.38, made of aluminum tubing, 0.63 cm. diameter, was cleaned and floated without trouble on the surface of pure water. The shape of the water surface at the air-water contact is shown in Fig. 4. The ring was then oiled slightly. The air-water contact angle was reduced, as shown in Fig. 5, to such an extent that it was impossible to float the ring. The same was true, as might be expected, when a cylinder of aluminum replaced the ring. A similar cylinder of glass tubing exhibited such a small air-water contact angle that it could not be floated.

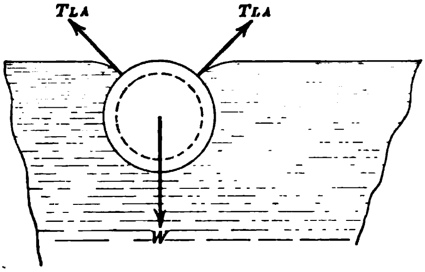

FIG. 4.

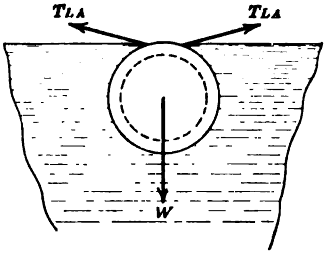

FIG. 5.

Adsorption

The surface layer between two physical phases is the seat of conditions of density and viscosity, also of apparent forces or energy manifestations, which are notably different from those in the bulk of either phase. On philosophical grounds it is impossible to consider that a real physical discontinuity occurs at the boundary between two media. In other words, there must be a very thin layer of transition in which there is a rapid but continuous change in the concentration of the components. This change in the concentration of a component at the interface is called adsorption, and may occur even between two phases which are ordinarily regarded as immiscible.

Adsorption at a gas-liquid interface may be demonstrated as follows: If a solid, which has been heated in a vacuum, is introduced into a measured volume of a gas over mercury in a calibrated tube, an amount of the gas will be adsorbed, as is shown by the change in pressure and volume compared to the space originally occupied.[1] These additional facts are established:

(*a*) The amount of the gas adsorbed at constant temperature increases with the pressure.

(*b*) It is different for different gases.

(*c*) It is different for different solids.

(*d*) It increases as the temperature decreases.

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[1] Freundlich: *Kapillar Chemie*, p. 92.

(*e*) There is an energy transformation which is indicated by the heat developed through adsorption, analogous to the Pouillet phenomena mentioned later.

(*f*) Chemical reactions are assisted by the adsorbed layer.

It follows that the gas layers must vary in density, falling off rapidly with increasing distance from the solid. Quincke assumes that the density of the gas next to the solid is equal to that of the solid and concludes that the amount adsorbed will increase with the density of the solid. From these facts we conclude:

1. That gases and solids exhibit selective adhesion and that, therefore, gas bubbles will attach themselves more persistently to some substances than to others.

2. That this selective adhesion is a manifestation of a definite amount of energy possessed by each unit area of a gas-solid contact, and that this potential energy is capable of variation.

3. That chemical reactions which diminish this potential energy are aided by adsorption.

Adsorption at a liquid-solid surface manifests itself in a vacuum, or where the vapor phase is negligible, by the way in which the liquid spreads or gathers itself together on the solid; in other words, in the way in which the liquid wets or adheres to the solid. It is further manifested by an evolution of heat, known as the Pouillet phenomenon. A calculation of the condensation necessary to evolve this amount of heat, in the case of water against glass, indicates that the specific gravity of water in the adsorbed layer is increased to about 2.1.[2]

Adsorption of the gas at a gas-liquid surface is indicated

1. By the effect on the surface tension. The surface tension of a freshly formed mercury surface does not change in a vacuum, but falls off in the presence of different gases for about an hour. Certainly the density of a liquid cannot be constant at the boundary but must go over continuously into that of the gas.

2. By the increase in the solvent power of the surface.[3]

3. In the case of contaminated liquids, by the concentration of one or more of the components of the liquid at the gas-liquid surface. Every unit area of such a boundary possesses a definite potential energy which always tends to a minimum. If, therefore, the surface tension of a solution depends upon the presence of any component, such a change of concentration of that component will occur as will reduce the potential energy, *i.e.*, the interfacial tension. In other words, any component which reduces surface tension will be found in excess at the surface of a solution. For example, the surface tension of water is greater than that of alcohol. Experimentally, a drop of alcohol on a thin film of water

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[2] Lewis: *Phil. Mag.*, 20, p. 502, 1910.

[3] Pockels: *Nature*, Mar. 12, 1891, p. 439.

rapidly reduces the surface tension of the water and the latter draws away from the alcohol. On the contrary, a drop of water on a thin film of alcohol spread over glass does not at once diffuse into the film but remains gathered in a heap. Such diffusion would increase the surface energy of the system, hence the water concentrates away from the gas-liquid surface. The greater viscosity of the surface of a solution above that of the bulk, or of that of a pure liquid, has long been recognized[4] by its damping effect upon a swinging magnetic needle and may properly be ascribed to gas-liquid adsorption. Closely connected with this is the formation of elastic solid skins or very viscous layers at a free surface, as, for example, in the case of solutions of peptone and dye stuffs. A peculiarity of saponine solution is the rigidity of its surface while the interior remains more mobile.[5] The surface of a freshly formed fairly concentrated solution of fuchsin is quite mobile, but in the course of a few hours it changes to a reflecting skin with solid properties. Similar results are obtained with methyl-violet and peptone. In the case of the latter substance the skin is highly elastic. There seems to be no doubt that true adsorption is present here. In the case of crystal-violet, which closely resembles methyl-violet in its properties, a solution of 1 g. to the liter lowers the surface tension from 75 to 69.9 dynes per centimeter. Other causes for the production of a solid layer may, however, be present, for many of these substances in concentrated solutions stiffen into gelatines and since the concentration of the contaminant is great at the surface and the solubility has also a different value, the solid remains persistently. Either a chemically irreversible change or a transformation into a more difficultly soluble phase at the surface is clearly the explana-

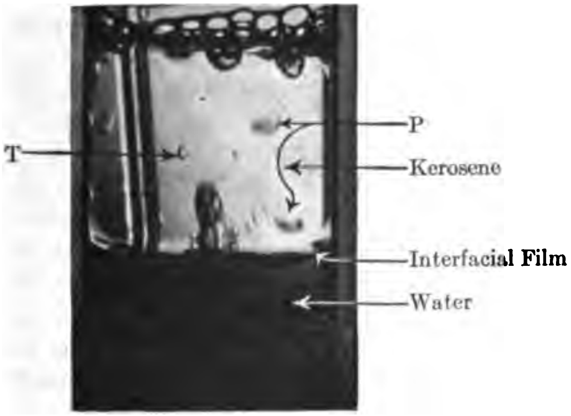

FIG. 6.

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[4] Daniells: *Physics*, p. 258, Fr. p. 76.

[5] Boys: *Soap Bubbles*, p. 115.

tion of the persistence of the froth in albumen solution and the like. The properties of such surfaces apparently pass over imperceptibly into those of colloids.

Adsorption at the boundary between two liquids is evidenced by the effect on the interfacial tension just as in the gas-liquid solution surface, although the process is not one usually described as ordinary solution and one may also have to reckon with chemical reactions in the transition layer. With liquid-liquid surfaces, as in gas-liquid surfaces, the adsorption frequently gives rise to very viscous layers. The presence of such a viscous layer at an oil-water interface is easily shown by pouring any clear oil, kerosene, liquid vaseline, etc., onto water and then bubbling a gas through the water. Such an experiment is shown in Fig. 6. The interface has all the appearance of an elastic skin. Bubbles rising through the water and striking the under side of the interface stretch the film (see *H*, Fig. 7), and rising farther drag away a mass of water surrounded by this viscous layer. The system now appears as shown at *M* and rises to the oil-air surface on account of its lower specific gravity. Here the film, together with the excess water carried up as shown at *C*, breaks away and falls back through the oil, not in spherical form, as would be the case were the water drop not surrounded by a viscous film, but in hemispherical form (see *P*, Figs. 6 and 7) often trailing behind it a film with ragged edges, as it broke from the bubble. The tadpole-shaped water drops, *T*, (Figs. 6 and 7) are further evidence of the high viscosity of the oil-water interfacial film.

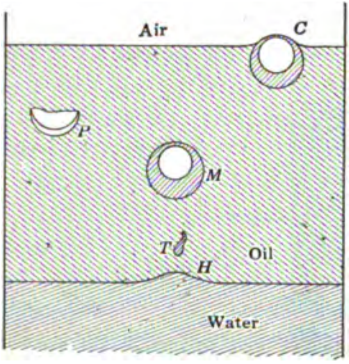

FIG. 7.

Viscosity

A marked increase in the viscosity of interfacial films is produced by the presence of finely divided solid matter. This increase is apparent in the experiment just described when finely powdered sulphide is thrown into the oil and allowed to settle to the interface, where it becomes entangled in the film. When gas bubbles are introduced, as before, the return water drops, coated with a film containing the solid particles, are much more irregular in shape than previously, and their coalescence after reaching the interface requires days or weeks. An even more convincing proof of the increase in viscosity of an interfacial film is given by the following experiment. If a needle is floated at the center of a surface of pure water in a beaker 4 in. in diameter and a chip of wood is floated near the wall of the beaker, the needle may be caused to revolve by means of a magnet without disturbing the chip. If the surface of the water is dusted over with fine ore, the whole surface together with the chip moves as though it were a rigid solid.

SUMMARY

The potential energy at a gas-sulphide contact is less than at a gas-gangue contact; hence gas bubbles will cling with greater persistence to sulphides than to gangues.

Oil replaces water at the surface of sulphide minerals.

Water replaces oil at the surface of gangue minerals.

Water replaces gas more readily at an oiled surface of a solid than at a clean surface.

The addition of any contaminant to water lowers the surface tension.

In any body of contaminated water there will be a concentration of the contaminant at the air-liquid surface.

Adsorption at a gas-liquid surface lowers the surface tension and increases the viscosity.

Adsorption at a liquid-liquid surface produces a film whose viscosity is higher than that of the bulk of either liquid.

The presence of finely divided solid matter in a film markedly increases the viscosity of the film.

APPLICATION TO COMMERCIAL FLOTATION PROCESSES

Film Flotation

Pulp with or without oil or acid is fed gently, at an acute angle, onto the surface of a body of still water. The sulphide floats and the gangue sinks.

Possible cases:

Case 1.—Sulphide, gangue, water.

Case 2.—Sulphide, gangue, water, oil.

Case 3.—Sulphide, gangue, water, acid.

Case 4.—Sulphide, gangue, water, acid and oil.

Case 1. *Sulphide, Gangue, Water.*—The governing factor in the initial flotation of the sulphide and immersion of the gangue is the difference in the air-water contact angle with the sulphide and gangue surfaces respectively. If the difference is great, as in the case with galena and quartz, good separation is obtained. After a considerable amount of sulphide has been floated, if the surface flow is sufficiently impeded, the particles congregate into clumps by the well-known phenomenon of apparent attraction of floating particles,[6] a scum is formed (whose viscosity and resistance to rupture are many times greater than that of the original water surface) and the floated particles are rendered more immune to immersion.

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[6] Hastings and Beach: *General Physics*, p. 156.

Case 2. *Sulphide, Gangue, Water and Oil.*—When oil is added in this process the phenomena are entirely different from the simple film flotation of Case 1. The oil concentrates at the surface since such concentration reduces the surface energy of the system. This adsorption of the oil at the gas-water surfaces causes the formation of a viscous film. When the mixture of sulphide and gangue is introduced at the surface, the sulphide particles tend to migrate into this layer and the gangue particles migrate into the water, for a sulphide particle in contact with oil represents the condition of least potential energy which is possible for the sulphide particle to assume in the system of oil, sulphide, and water. Likewise the gangue particle surrounded wholly by water represents the condition of least potential energy for the gangue particle to assume in this system. In this case also, the formation of a scum of floated sulphide increases the stability of the float.

Case 3. *Sulphide, Gangue, Water and Acid.*—The effects of acid are: (*a*) to diminish the surface tension of the liquid, (*b*) to diminish the gas-liquid contact angle, and (*c*) to increase the viscosity of the gas-liquid surface. The diminutions of (*a*) and (*b*) are more marked with gangue minerals than with sulphides. The result is that cleaner concentrate may be expected than in either of the previous cases, but at the expense of richer tailing.

Case 4.—*Sulphide, Gangue, Water, Oil and Acid.*—In this case a combination of results such as can be predicted from the preceding cases is obtained.

Froth Flotation

In order to explain froth flotation it is necessary and sufficient that the gas of each bubble shall be inclosed by a film of contaminated water which shall possess the following characteristics:

1. Low surface tension.
2. High viscosity.
3. A variable concentration of the contaminant (reagent).
4. A preferential adhesion of the bubble film to the sulphide mineral compared to that for the gangue minerals.

We will examine first the conditions required for the formation of a froth, or the continued existence of a thin film. Solutions which form froth are preëminently aqueous solutions and the properties of the liquid film are only secondarily determined by those of the gas.

The durability of a liquid film depends upon one or more of the following conditions:

1. A low surface tension which is locally variable so as to produce stable equilibrium.

2. High viscosity, which may pass over into
3. Chemical irreversibility and the production of solid skins.

Pure liquids do not foam; for example, water or alcohol. The reason is obvious. As the film is thinned out by stretching or by draining away of the liquid, the surface tension is reduced at some part below the general constant value. As soon as this begins the thicker and more powerful parts of the film drag away from the weakened parts which at once completes the rupture. These inequalities evidently would not be so marked or rapid in their operation if the surface tension were low. Increase of viscosity would also slow up the process. High viscosity and low surface tension do not occur in pure liquids. But the most important condition for durability is some means by which the equilibrium of the forces at any point in a film may be restored, when a variation of some of the forces occurs. In the case of a compound liquid or solution this is effected by the adsorption or change of concentration of one or more of the components in the film.

The surface tension of a solution is in general notably different from that of the pure solvent, and in case of water, whose surface tension is the greatest of any liquid with which we are concerned, even a minute trace of impurity is sufficient to lessen its surface tension considerably.

Consider a film of water stretched on a vertical ring of wire. If the surface tension remained constant, as it does in the pure liquid when the thickness exceeds 0.000001 cm., the weight of the lower part would stretch down the upper part until it broke. If the water contains some component whose depletion at the weaker points increases the surface tension, equilibrium will be preserved. In the stretching of a film, and in the general running away of the liquid between the surfaces of the film, which reduces the total available amount of the contaminant, such decrease of the concentration and increase of surface tension does occur, and the film remains stable under a considerable variation of external conditions. The formation of bubbles as a result of this variation of surface tension alone is well exemplified by a simple aqueous solution of soap or of acetic acid. The running out of the liquid between the two surfaces is greatly retarded by the viscosity of the liquid, a property which may be largely influenced by the surface adsorption of one or more of the components.

When, then, gas bubbles are introduced into a liquid pulp where oil is present there is formed about each bubble a liquid film whose surface tension is less and whose viscosity is greater than that of the bulk of the liquid. Some of the solid particles of the pulp move into the film and are raised to the surface with the bubble. Since there is a concentration of oil in the film, and since the diminution in potential energy at an oil-sulphide contact is greater than at a water-sulphide contact, the contaminated layer replaces the water on the sulphide surface and the sulphide moves into the bubble film, while the gangue, on which water displaces oil, remains in greater measure in the body of the pulp. The bubbles, therefore, as they arrive at the surface, carry an excess of the sulphide minerals. Upon their arrival at the surface, the bubbles of the contaminated liquid persist, owing: (1) to their lower surface tension; (2) to their ability to adjust this tension to a state of stable equilibrium; and (3) to their greater viscosity which is markedly increased by the presence of the solid particles.

Mechanical-Agitation Froth Process

Sulphide and gangue minerals are beaten up with water and oil, with or without acid, then allowed to flow into a box containing a considerable body of liquid nearly at rest. Bubbles coated with a preponderance of sulphide particles float to the surface and form a heavy, persistent froth. The gangue particles sink.

Two cases arise:

Case 1.—Sulphide, gangue, water and oil.

Case 2.—Sulphide, gangue, water, oil and acid.

Case 1. *Sulphide, Gangue, Water and Oil.*—When this pulp is beaten, air is mechanically entrapped in the form of bubbles. At the surface of every bubble in the mass there is a gas-contaminated-liquid contact which results in the adsorption at this surface of the contaminant, oil, and the production of a viscous film into which the sulphide particles, circulating in the mass, pass with a diminution in the potential energy of the system. The result is that in a very short time after the air bubble is entangled in the pulp, it is surrounded by a viscous sheath composed of an oil-water interfacial film in which are entangled a large number of sulphide particles. The presence of the solid particles greatly increases the viscosity of the bubble sheath. When the solid-coated bubble arrives in the settling box or spitzkasten it rises to the surface. Here the bubble persists, or, bursting, transfers its load to other bubbles. This bubble persistence is due to a combination of several factors. The oils used have, in general, a slower evaporation rate than water. The tension of the bubble film is lower than the tension of a pure water bubble. The bubble has the power of adjusting itself to its tension, within limits, without bursting. The presence of the large amount of solid matter enormously increases the viscosity of the film.

Case 2. *Sulphide, Gangue, Water, Oil and Acid*—The addition of acid has the twofold effect of further lowering the surface tension and increasing the adhesion ratio $\frac{\text{oil-solid}}{\text{water-solid}}$. The result is, in general, cleaner concentrate with or without an increase in the sulphide content of the tailing.

Heating the pulp has, in some cases, a beneficial effect. Where this is true it is probably due to (*a*) decreased surface tension and consequent increased stability of the froth; (*b*) increased number of air bubbles formed by the air released from solution; (*c*) in the case of viscous oils, the greater area over which oil is spread and consequently the greater number of bubbles with a viscous oil-water interfacial film; (*d*) probable increase in solubility of the oil and consequent greater diffusion, resulting in more widespread adsorption in bubble films.

Pneumatic Froth Process

Sulphide and gangue minerals mixed with water and oil, with or without acid, are run into a tank with a porous bottom through which air is forced. The air bubbles rise to the surface with a coating of solid particles, preponderantly sulphide, while the gangue particles sink.

The principles involved in this method are the same as explained in the agitation-froth process. The only difference is in the method of introducing air. The result of this difference is that the bubbles in the pulp are much larger than in the agitation froth method; they arrive at the surface less heavily loaded in proportion to their area; the bubble films are, therefore, less viscous, and the froth less persistent.

Potter-Delprat Process

Sulphides and carbonates, with or without other gangue minerals, are treated with hot, dilute sulphuric acid. Bubbles of carbon dioxide and hydrogen sulphide are formed which rise to the surface with a sulphide coating and there form a froth. That part of the gangue not dissolved remains immersed. In this method as in the other froth methods, gas bubbles are formed which are surrounded by films of contaminated water, the contaminants in this case being sulphuric acid, lead sulphate, calcium sulphate and other salts formed by the action of the sulphuric acid. The films have a higher viscosity and a lower surface tension than is possessed by the bulk of the liquid. The sulphides move into the bubble films because the system composed of sulphide and this contaminated layer has a lower potential energy than the system composed of the pulp in the bulk of the liquid. The writers at first suspected that the selective action in this case might be due to preferential gas adsorption at the gas-sulphide contact, as opposed to a gas-gangue contact, but microscopic examination of mineral froths collected from the process showed that the solid particles in the froth were completely within the films and at no point in contact with gas. The persistence of the froth is due to the factors explained in connection with the other froth processes.

While the writers have made no appeal to electrostatic forces or to colloidal phenomena in this discussion, they realize that the potential energy existing at the contact of dissimilar substances may well include electrical forces and that migration of the suspended solid particles under the influence of electrical charges, similar to the migration of colloids, may account for some of the selective action of the bubble films. But the agitation of the pulp in the mechanical- and pneumatic-froth processes and the generation of carbon dioxide gas on carbonate particles in intimate contact with sulphides in the acid-froth process, are sufficient, in their opinion, to bring every sulphide particle into contact with a bubble film. Once in contact, the preferential adhesion of the contaminated layer to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films, while on the other hand, the replacement of gas or oil by water on the surface of gangue particles explains the wetting and continued immersion of the latter.

The writers have a considerable bulk of experimental data on which many of the statements in the foregoing explanation are based. These, together with the data from other experiments which are projected, and photographs of many of the phenomena mentioned, they hope to present in a later paper.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Some Miscellaneous Wood Oils for Flotation.*

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(Arizona Meeting, September, 1916)

THE testing of flotation oils has occupied a large part of the time of the testing departments of various companies using the flotation process in the beneficiation of their ores. The great differences in ores and in oils has made such work necessary as it has not been possible to select the oil best suited for the purpose without trial of many kinds. In fact, individual shipments of flotation oils from the same manufacturer will often differ so radically that the mill men have been forced to test every lot of flotation oil before its use in the mills.

Further, although almost every imaginable product of the distillation of wood, from alcohol to pitch, has been tested in a more or less perfunctory manner by various people, few companies have allowed the results of their tests to be made public.

This work was undertaken for the purpose of obtaining reliable data regarding the value as flotation agents of a number of products that are obtained from the distillation of wood by different methods, and incidentally to determine whether some of the products not used at present could be used. Large quantities of such materials are being wasted or burned in the wood-distillation plants of the United States, and if some use other than fuel could be found for these materials, such a use would represent a distinct saving to the country as a whole, would increase the profit of wood-distilling plants now operating and perhaps make it possible for others to operate which at present cannot do so except at a loss. At the same time it was thought that if such products can be used as flotation agents, then their use would serve to increase the number of oils and other products available for flotation purposes. It was also hoped that it might be possible to discover a substitute for some of the most effective but comparatively high-priced oils which are now used for flotation purposes. Hence these fractions occupy the major portion of the following list.

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* By permission of the Director, U. S. Bureau of Mines.

This work is the result of coöperation between the Forest Products Laboratory of the Department of Agriculture, the Metallurgical Research Department of the University of Utah, and the U. S. Bureau of Mines. The authors are members of the three organizations.

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In taking up this work, the Forest Products Laboratory of the Department of Agriculture, which is interested in finding uses for byproducts from wood distillation, was represented by R. C. Palmer. The Bureau, which is interested in finding supplies of suitable flotation oils, was represented by O. C. Ralston, and the Department of Metallurgical Research of the University of Utah, which is doing coöperative work with the Bureau of Mines, was represented by G. L. Allen.

Three ores were chosen for tests of the series of oil samples herein described.

Ore A was from the mine of the Arthur Zinc Mining Co. near Ruby Valley, Nev. It was chosen on account of its simplicity. It consisted mainly of galena, sphalerite and quartz minerals, which were crystallized in such a way that they could be broken away from each other by crushing to the size necessary for flotation. Ground quartz was mixed with it to make it sufficiently low-grade. Any marked tendency of one mineral to float in preference to another could be observed with such a "free" ore. Consequently, the value of the oil for differential flotation of mixtures of galena and sphalerite could be observed; also the conditions which allowed the best extraction or the cleanest concentrate of any of these minerals with any given oil.

Ore B was the heavier sulphide ore from Cripple Creek, Colo. This siliceous ore contains some pyrite, some gold and silver as the tellurides, sylvanite and calaverite, and other tellurium minerals. The tests on this ore would allow observations on the flotation of pyrite as well as of the famous Cripple Creek gold telluride minerals. Wherever the pyrite contains gold it would be desirable to float it, while barren pyrite ought to be kept with the gangue. Hence the tests should be carried on with an ore containing both types of minerals, as was done in this case. This sample was furnished by the Golden Cycle Mining & Milling Co.

The ore C was the Utah Copper Co.'s milling ore from the mine at Bingham. A great many copper minerals are present in this ore—chalcopyrite, cupriferous pyrite, etc. The gangue is siliceous, and the ore is well known. It is a hard ore to treat by flotation and nearly any oil that will treat it is almost sure of success with most other copper ores. One peculiarity of the Utah Copper ore is that it is much easier to float in an alkaline solution.

Each of the ores was tested with three different tests—an acid test, an alkaline test, and a test run in neutral water. The water used in all the flotation tests was the Salt Lake City water. It contains a small amount of lime so that the water used in the "neutral" test was in fact slightly alkaline. This method of testing was adopted for the reason that it is known that some oils will refuse to work at all in neutral solution although they do good work in acid solution. Others are more successful in alkaline solutions. The most advantageous amounts of acid or of sodium carbonate were not determined. The procedure followed, in order to facilitate work, was to add the reagent drop by drop to the charge in the flotation machine until it could be seen that it was causing some difference in the appearance of the froth or of the gangue. Further additions of the reagent were often made after most of the froth resulting had been skimmed off, and it frequently happened that a further amount of excellent froth would result.

Although such a method of obtaining the best conditions is not satisfying to certain types of mind, and is very aptly characterized as the "American" type of research, it is practically necessary to follow such a method in order to get anywhere with such testing. So many other variables are involved that to painstakingly follow out each one for only one oil, while the others are kept constant, is an immense undertaking. A flotation testing man must be an opportunist and seize on the best conditions for flotation as he is able to see them in his machine when varying the conditions, such as amount of agitation, depth of pulp in machine, amount of oil, amount of acid or alkali, temperature, etc.

The Janney test machine described in the *Mining and Scientific Press* for Jan. 1, 1916, was used in all this work. Our reason for using this machine is that it permits of quick work and is so designed that practically every particle of flotation concentrate and tailing can be recovered and weighed, while many other machines have inaccessible corners where some of the material escapes.

The extractions were calculated by the following well-known formula which does not require weights of the products involved.

$$\text{Extraction} = \frac{100\,(h - t)c}{h(c - t)}$$

where the analyses of the head, or original ore, tailing, and concentrate are represented by the letters in the equation. This formula assumed that any middling produced can be separated into concentrate and tailing.

Each test was "roughed" and "cleaned." The rough tail was final but the concentrate was poured back into the machine with more water and a clean concentrate and middling produced.

The oils tested in these experiments were furnished by the Forest Products Laboratory,[1] Madison, Wis. Most of them were either commercial samples or were produced in the laboratory in semi-commercial apparatus under conditions entirely comparable with plant practice.

The following is a somewhat detailed description of the different oils.

No. 23 was an authentic commercial pine oil, specific gravity 0.9385 at 21° C., with a sulphonation residue of 0.8 per cent. The oil was produced by the steam distillation of fat southern pine wood, the crude oil being redistilled to separate the turpentine (boiling point 155 to 170° C.), from the pine oil (boiling point 190 to 220° C.). As there is some confusion in the mining industry in regard to the name "pine oil," many other oils produced from dead pine wood also being known as "pine oil," it may be well to describe the oil to which the name is correctly applied. There is a resin in the sapwood of the living pine, which flows out when the tree is wounded. This resin is the source of the ordinary turpentine and rosin of commerce. When the sapwood turns to heartwood, the resin becomes hard and during the ageing process another oil is formed which did not exist in the sapwood. This oil is heavier than turpentine, has an entirely different composition, and a much higher boiling point, and it is this oil that is correctly termed "pine oil." This oil can be removed from the dead pine wood either by the action of steam or dry heat. Dry heat carefully controlled so as not to exceed the maximum boiling point of the oil will remove it in practically as pure condition as when the wood is treated with saturated steam. Pine oil is produced either as a yellow or water-white oil, the former being produced in the first separation of the crude turpentine-pine-oil mixture, and the latter by redistillation of the yellow oil. If the wood is heated at a much higher temperature, as it is when the temperature is not controlled during the early stages of the destructive distillation process, the natural oils (turpentine and pine oil) in the dead wood are contaminated by tar and tar oils produced by the decomposition of rosin and wood. It is practically impossible commercially to separate a pure pine oil from this mixture.

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[1] Maintained by the Forest Service, U. S. Department of Agriculture in coöperation with the University of Wisconsin.

Chemically, pine oil has been found to be composed chiefly of a terpene called terpeneol, although other terpenes such as borneol, fenchyl alcohol, and cineol, are also present. Since the detection of mineral oils as adulterants in pine oil is of special interest it may be well to point out that the sulphonation test may be successfully used for the detection of mineral oil. A pure pine oil should give a sulphonation residue of less than 1 per cent. Special care must be taken, however, in carrying out the test, and it is suggested that the test be made as follows:

Take 25 c.c. of 37 normal sulphuric acid in a Babcock bottle and add 5 c.c. of the oil to be tested, 1 c.c. at a time, shaking vigorously after each addition. When the oil is all added, place the bottle in a water bath and heat at 100° for 1 hr., shaking cautiously from time to time to insure a perfect mixture of the oil and acid. Cool and fill the bottle with ordinary sulphuric acid and either allow it to stand for 24 hr. or centrifuge it in order to read the residue. It is absolutely essential to have the strength of the acid exactly 37 normal; the cooking of the mixture for 1 hr. is equally important for securing low residue with pure pine oil. The index of refraction of the residue is also of value. With pure pine oil the residue should have an index of over 1.5, while a mineral oil will have a residue with an index of less than 1.48.

A pine oil was included in these tests for the purpose of comparing an oil of known flotation value with the other oils. The sample had been shaken repeatedly with hot water in order to remove any water-soluble constituents (probably the fenchyl alcohol), to determine whether the removal of water-soluble material had any noticeable effect on the value of the oil for flotation. About 5 per cent. was found to be soluble in water.

No. 11 was a commercial wood oil produced in the destructive distillation of hardwood, the particular species in this case being maple and birch. The specific gravity of the oil was 0.934 at 21° C. The chemical composition of this oil is unknown, although it probably contains a number of complex ketones. In the destructive distillation of hardwood a heavy tar settles out of the crude pyroligneous acid, the watery distillate containing the wood alcohol and acetic acid, and this tar has included in it some of the pyroligneous acid. To remove this acid the tar is usually placed in a wooden still and either washed by blowing in exhaust steam or is simply heated with steam in closed coils. In either case a watery product distills over and with it is carried this light tar oil or "wood oil," as it is called in the plant. This oil is practically a waste product and is burned for fuel in most plants. It is an excellent solvent, however, and a small amount is sold for that purpose. A cord of hardwood gives on an average about 2 gal. of this oil and the production in all the hardwood distillation plants is probably close to 2,500,000 gal. annually. It could probably be bought for $0.10 to $0.15 per gallon f.o.b. plant, as this is the price usually paid for it in the crude condition for a solvent.

No. 18 was a crude hardwood tar just as it settled out of the pyroligneous acid, as described above. This tar was produced in a semi-commercial laboratory distillation plant by the distillation of equal proportions of beech, birch and maple. It still contained the pyroligneous acid and wood oil. On distillation the sample gave 12 per cent. pyroligneous acid, 8 per cent. wood oil, 42 per cent. wood creosote oil, and 38 per cent. wood pitch. The specific gravity of the combined wood oil and wood creosote was 1.067 at 21° C. When the distillation of the crude tar is continued with direct heat after distilling off the watery distillate and light oil (wood oil) the oil obtained is similar to the hardwood creosote now on the market for flotation. The object in testing a crude hardwood tar of this kind was primarily to determine to what extent the pitch present in the tar acted as a deleterious adulterant. Excluding the wood oil, about 15 gal. of this tar is produced per cord of hardwood, making available per year over 20,000,000 gal. of this crude tar. This tar with the wood oil and pyroligneous acid removed is generally used for fuel at the distillation plant, where it is worth about $0.03 per gallon for this purpose, in comparison with the normal prices for other kinds of fuel. It could probably be bought for $0.05 to $0.06 per gallon f.o.b. plant.

No. 15 was an entirely different tar product produced in the distillation of hardwood. After the pyroligneous acid has been allowed to stand to settle out the oily tar, it contains a tar-like substance dissolved in it which has to be removed before the liquor can be worked up into alcohol and acetic acid. Since this tar is non-volatile it is removed by completely distilling the pyroligneous acid, the tar being left as a residue in the still. This sample was produced in the laboratory distillation plant from beech, birch and maple. It is a heavy, non-volatile, viscous, tar-like substance, specific gravity about 1.4. Its composition is unknown. It is about 50 per cent. soluble in water, the remainder dissolving in the pyroligneous acid because of the solvent action of the alcohol and acid constituents. The yield of this "dissolved tar" or "residual tar" or "acid water tar" or "copper still tar," which are the various names given to it in the plants, is from 6 to 8 gal. per cord, making a total production in the United States of about 9,000,000 gal. per year. It is strictly a waste product at present, although worth about $0.025 per gallon as fuel. It could probably be obtained for $0.04 to $0.05 per gallon at the plant.

No. 16 is a 15 per cent. water solution of the water-soluble portion of No. 15, just described. The specific gravity of the solution was 1.061 at 21° C. This sample was prepared to determine the value of the water-soluble portion as compared to the whole tar.

There have been many references in the literature to the use of pyroligneous acid for flotation, but no data have been given showing the actual value of this material. These tests included several samples of pyroligneous acid to determine this point.

No. 14 was a semi-commercial pyroligneous acid produced by distilling maple. The specific gravity of the sample was 1.036 at 21° C. It contained 9.7 per cent. dissolved tar, 4.8 per cent. wood alcohol, 11.9 per cent. acetic acid. The value of this liquor for its alcohol and acid content is over $0.05 per gallon.

No. 12 was a commercial pyroligneous acid from a plant distilling birch and maple, from which the dissolved tar had been removed by distillation. The specific gravity of the sample was 1.008 at 21° C. It contained 3.1 per cent. wood alcohol and 5.9 per cent. acetic acid, making it worth a little over $0.03 per gallon for its content of these products.

No. 10 was commercial crude pyroligneous acid produced in the destructive distillation of southern yellow pine. The specific gravity of the sample was 1.034 at 21° C. It contained much less alcohol and acetic acid than the hardwood pyroligneous acids, the composition being 1.8 per cent. alcohol, 2.85 per cent. acid and 7 per cent. dissolved tar. Except under abnormal conditions it does not pay to refine pine-wood pyroligneous acid in competition with the hardwood product and it is thrown to waste. This particular sample would not be worth over $0.0175 per gallon for the alcohol and acid in it. A cord of pine (4,000 lb.) will yield from 125 to 150 gal. of crude pyroligneous acid. It would certainly not pay to buy the crude acid at even $0.02 per gallon f.o.b. plant on account of the freight charges on the large volume of water in it. The concentration of the crude distillate would simply give a thick tar containing a small amount of the acid and none of the alcohol, but if the tar is the valuable constituent this would seem to be the only feasible method of marketing softwood pyroligneous acid for flotation.

The Results

The results of the flotation of these three ores with the above set of oil samples are contained in Tables 1 to 3. In Table 1 the analyses of the concentrate and tailing for each test are given. In the other two tables only the analyses of the concentrate samples are given. Accompanying the analyses of the concentrate samples in each case are the percentages of extraction obtained. The test in neutral, acid and alkaline solutions for each oil is given.

It can be seen that many of the oils give fully as satisfactory results as pine oil for practically all the minerals tested. Oils 11, 18, and 15 are the ones of most interest on account of being immediately available at low prices and because they can be used in small amounts with success. For all-round work, No. 18 seems to have given good extractions of all the metals tested, together with fair grades of concentrate. It will be remembered that this is the crude hardwood tar which has settled out of the pyroligneous acid.

It is a pleasant surprise to find that No. 16, the 15 per cent. solution of No. 15 in water, is so efficient in comparison with No. 15. Here is something which is really not an oil because it is completely soluble in water. It is satisfactory for all the minerals except the copper minerals of ore C.

It will be noticed that the best extractions of the zinc, and generally the highest-grade zinc concentrates, were obtained in solutions that had been acidified. Differential flotation of galena in the presence of sphalerite is seen to be most marked in alkaline or in neutral solutions. No. 14, the hardwood pyroligneous acid, seems to be best adapted for this.

The extractions of the gold in the Cripple Creek ore and the grades of concentrate are surprisingly good. Oils 18, 15 and 16 seem to be capable of producing the highest-grade concentrates. By further treatment of the tailing it should be possible to get even higher extractions, although the grade of the total concentrate would be cut down. No attempts to run tests in this manner were made, as the object of the work was the finding of oils best adapted to do certain things. The relatively high

TABLE 1.—*Tests on Ore A in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Lead, 3.4 Per Cent.; Zinc, 4.74 Per Cent.

Oil No.	Character of Oil	Remarks	Total Time, Min.	Total Oil, Lb. per Ton	Total Acid, Lb. per Ton	Total Alkali, Lb. per Ton	Lead		Zinc	
							Per Cent.	Per Cent. Extraction	Per Cent.	Per Cent. Extraction
23	Pine oil...............	Concentrate	10	0.375	....	...	55.9	53.1	12.4	62.2
		tailing.	10	0.375	....	...	1.6	....	2.3	
			10	0.375	11.14	...	32.3	87.9	37.9	78.2
			10	....	3.36	...	0.4	....	1.1	
			15	0.375	....	8.0	18.0	89.5	25.5	79.1
			10	....	....	2.0	0.4	....	4.4	
11	Hardwood "wood oil"..	....	17	1.11	....	...	57.2	92.6	20.4	58.2
			10	0.18	....	...	0.3	....	2.3	
			16	0.94	3.68	...	37.9	88.7	34.2	72.6
			15	0.17	3.68	...	0.4	....	1.4	
			15	1.29	....	4.0	48.0	97.2	32.0	64.8
			5	....	....	...	0.1	....	1.8	
18	Crude hardwood "settled tar."	....	17	1.35	....	...	48.6	97.2	28.0	68.4
			3	....	....	...	0.1	....	1.7	
			19	1.24	3.68	...	37.7	98.2	33.4	76.5
			6	....	....	...	0.1	....	1.2	
			20	1.70	....	2.0	48.7	98.0	27.9	63.4
			4	....	....	...	0.1	....	1.9	
15	"Dissolved hardwood tar."	....	20	1.44	....	...	59.2	79.4	16.2	30.2
			7	....	....	...	0.7	....	3.5	
			15	1.44	5.52	...	32.8	88.7	13.0	69.8
			10	0.72	3.68	...	0.4	....	1.9	
			20	1.44	....	4.0	66.7	73.0	38.1	30.3
			8	....	....	...	1.0	....	3.4	
16	Water solution of No. 15	....	13	1.80	....	...	55.1	90.2	22.0	43.7
			5	....	....	...	0.4	....	2.9	
			19	1.80	7.36	...	30.3	92.2	40.5	77.2
			4	....	....	...	0.3	....	1.2	
			19	1.80	....	4.0	53.7	77.4	23.3	44.7
			4	....	....	...	0.8	....	2.9	
14	Crude hardwood pyroligneous acid.	....	20	14.00	....	...	54.3	86.2	22.7	51.5
			3	....	....	...	0.5	....	2.6	
			15	14.00	5.5	...	29.5	72.2	41.5	70.0
			5	....	....	...	1.0	....	1.5	
			17	12.0	....	4.0	62.0	54.8	9.8	42.7
			6	....	....	...	1.5	....	3.4	
12	Tar-free hardwood pyroligneous acid.	....	13	20.0	3.68	...	23.5	42.7	31.7	41.4
10	Crude softwood pyroligneous acid.	....	17	10.0	....	...	53.8	73.3	23.9	41.3
			13	4.75	....	...	1.0	....	3.0	
			18	12.0	3.68	...	31.6	64.7	35.2	57.3
		No cleaner	7	4.00	....	....	1.3	....	2.2	
		test.	12	16.00	....	4.0	49.1	57.8	17.9	15.0

grade of the heads in the tests on the Cripple Creek ore leaves tailing in all these tests too high to compare favorably with roasting and cyaniding. Hence, further work in adaptation of these oils to give higher extractions would have to be done. This is a question of mechanical manipulation with which every flotation experimenter is familiar. The extractions of the iron in the Cripple Creek ore are very poor with all the tests. As the ore was ground dry in a laboratory sample pulverizer this is not surprising, for pyrite is peculiarly sensitive to oxidation during such grinding and will not float well. Not being an important mineral in this case, the method of preparation of the sample did not matter. It is interesting to note that No. 11 and No. 10, the "wood oil" and the softwood crude pyroligneous acid, respectively, were the two oils which gave best extractions of the iron.

As was expected, the Utah Copper ore usually floated best in alkaline solutions, oils 11, 18 and 15 doing the best work.

Further work is being done, especially in the testing of the residues left by concentrating the softwood pyroligneous acid by evaporation, and many other oils are being tested in this manner.

TABLE 2.—*Tests on Ore B in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Gold, 0.608 Oz. per Ton; Iron, 5.2 Per Cent.

Oil No.	Total Time, Minutes	Total Oil, Lb. per Ton	Total Acid, Lb. per Ton	Total Alkali, Lb. per Ton	Gold[a]		Iron[a]	
					Oz. per Ton	Per Cent. Extraction	Per Cent.	Per Cent. Extraction
23	21	0.375	0.00	0.0	4.50	81.5	8.0	10.2
	21	0.575	3.68	0.0	6.42	78.7	9.0	16.5
	21	0.375	0.00	1.0	4.76	79.7	8.2	9.9
11	24	1.11	0.00	0.0	4.72	82.5	8.5	31.9
	22	1.11	3.68	0.0	6.12	75.8	17.2	41.1
	23	1.11	0.00	4.0	5.64	82.3	13.4	14.8
18	22	1.8	0.00	0.0	10.34	78.2	13.2	17.7
	21	1.5	3.68	0.0	11.52	78.0	14.3	19.6
	22	1.5	0.00	1.0	11.60	81.2	13.1	6.2
15	23	2.88	0.00	0.0	5.06	82.3	5.7	0.0
	21	2.88	3.68	0.0	13.80	74.8	9.6	11.8
	21	2.88	0.00	1.0	5.80	76.0	7.4	6.2
16	21	2.4	0.00	0.0	10.10	81.3	8.6	31.5
	21	2.4	3.68	0.0	12.60	81.2	....	0.0
	21	2.4	0.00	1.0	7.88	78.7	11.7	27.3
14	21	12.0	0.00	0.0	5.42	79.1	9.2	32.5
	22	12.0	3.68	0.0	7.22	81.7	12.0	40.3
	19	12.0	0.00	1.0	7.28	78.7	12.1	26.9
12	15	18.0	0.00	0.0	5.88	76.0	7.7	32.3
	20	20.0	3.68	0.0	6.76	81.7	15.0	12.8
	15	24.0	0.00	1.0	9.44	77.0	6.9	14.0
10	23	16.0	0.00	0.0	5.40	88.7	6.1	40.8
	21	16.0	3.68	0.0	4.98	79.6	6.0	41.2
	24	16.0	0.00	1.0	6.10	79.2	6.5	16.7

[a] In concentrate samples.

TABLE 3.—*Tests on Ore C in Neutral, Acid and Alkaline Solutions with Different Oils*

Heads: Copper, 1.58 Per Cent.

Oil No.	Total Time, Minutes	Total Oil, Lb. per Ton	Total Acid, Lb. per Ton	Total Alkali, Lb. per Ton	Copper[a]	
					Per Cent.	Per Cent. Extraction
23	17	0.375	0.00	0	3.83	
	16	0.375	7.36	0	8.61	28.7
	17	0.375	0.00	4	1.53	
11	20	0.92	0.00	0	15.95	29.3
	21	1.48	7.36	0		
	21	1.29	0.00	6	16.15	32.6
18	20	1.8	0.00	0	16.45	51.2
	20	1.5	7.36	0	2.48	
	20	1.2	0.00	4	22.27	68.2
15	21	1.44	0.00	0	5.84	4.3
	21	1.44	7.36	0	2.77	
	23	1.44	0.00	4	18.17	23.1
16	17	2.1	0.00	0	4.50	33.6
	18	2.7	7.36	0	1.72	5.6
	23	2.4	0.00	8	7.94	3.9
14	20	5.0	0.00	0	2.29	
	18	5.0	7.36	0	1.43	
	19	7.0	0.00	4	12.90	13.5
10	19	8.0	0.00	0	1.24	
	19	10.0	7.36	0	1.43	
	12	8.0	0.00	7	2.29	25.3

[a] In concentrate samples.

TABLE 4.—*Screen Analyses*

Ore A	Opening	Mesh	Weights	Assays	
	Mm.		Per Cent.	Pb, Per Cent.	Zn, Per Cent.
Through..........	0.589	28	100.00		
On...............	0.417	35	0.43	0.00	0.00
On...............	0.208	65	12.88	0.05	0.98
On...............	0.147	100	34.46	0.07	1.37
On...............	0.104	150	23.76	0.07	2.24
On...............	0.074	200	10.59	2.19	6.46
Through..........	0.074	200	17.75	19.08	14.60
Ore B					
Through..........	....	200	100.00		
Ore C					
Through..........	0.295	48	100.0		
On...............	0.208	65	2.2	1.58	
On...............	0.147	100	16.7	1.58	
On...............	0.104	150	16.0	1.82	
On...............	0.074	200	53.3	1.46	
Through..........	0.074	200	11.8	1.73	

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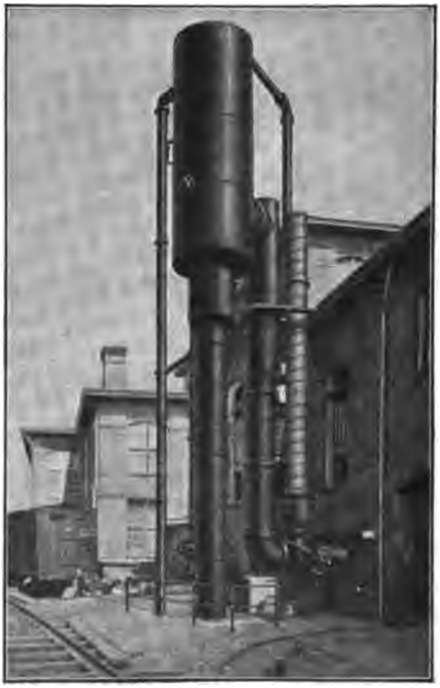

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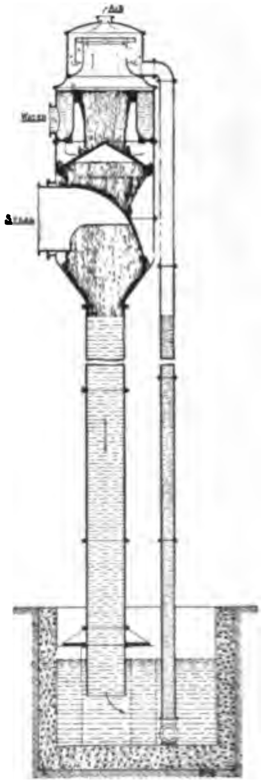

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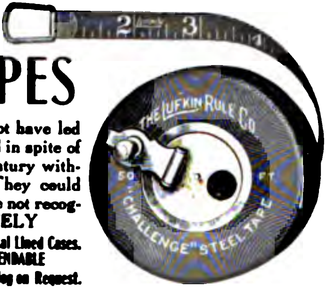

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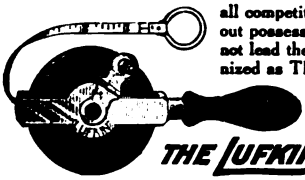

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According to statistics compiled by the U. S. Bureau of Mines, there were slightly more than 288,000 lb. of permissible explosives used in American mines in the year 1903, and the fatalities during that year, directly traceable to explosives, were 0.339 per 1,000 men employed.

In 1914 over 15,000,000 lb. of permissible explosives were used and the fatality rate dropped to 0.096 per 1,000 men employed. A reduction of 72 per cent. Again, in 1870 there were 13.47 fatal accidents for every million tons of coal mined in this country and in 1914 only 4.78 for every million tons mined.

There is still room for an even better showing than the above, because, compared to European statistics, we are still behind in the matter of safety and protection in coal mining.

The Bureau of Mines Statistics also show that of all the accidents due to explosives in coal mines 25 per cent. are caused by premature blasts caused by miners using short fuse or defective squibs.

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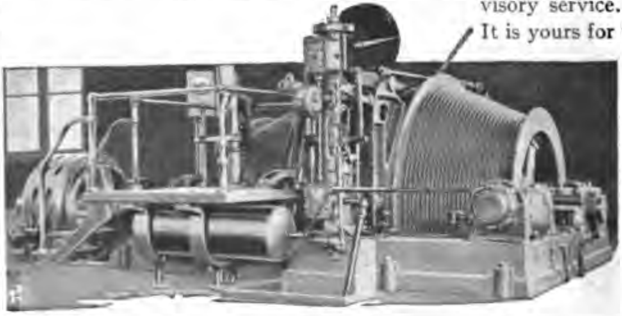

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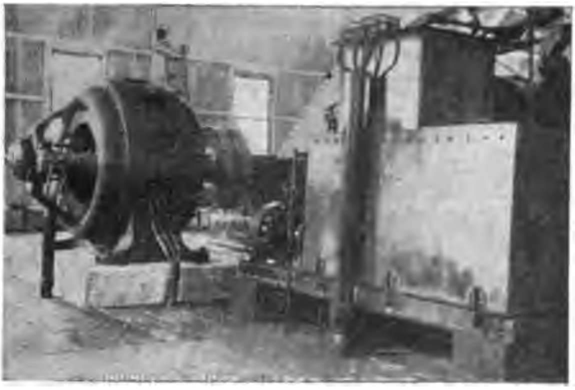

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SEPTEMBER, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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Signature of Applicant: ______________________________

Dated ______________________________ ***191***

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 117 SEPTEMBER 1916

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Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The Committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committee they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before Nov. 1, 1916, will, as far as possible, be published in the January Bulletin or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the Bulletin which goes out immediately preceding the meeting.

CHARLES KIRCHHOFF.

CHARLES KIRCHHOFF—IN MEMORIAM

Born, March 28, 1853 Died July 23, 1916

On July 25, 1916, the Board of Directors adopted the following minute and directed that it be entered upon the records of the Board and published in the Bulletin of the Institute, and that copies thereof be sent to the family of Mr. Kirchhoff:

Charles Kirchhoff was one of the oldest, most widely known and most highly esteemed members of the American Institute of Mining Engineers. On his return in 1875 from a European course of study, to begin a highly successful professional and business career, he joined the Institute, then just commencing to acquire both national and international recognition. He became at once, and for more than 40 years continued to be, its earnest and active supporter, serving it as Manager, Councilor or Director for terms aggregating 9 years, as Vice-President for 2 years, and twice as President for a year.

During the critical period of the erection of the United Engineering Society's building, and the transfer of the Institute to permanent headquarters therein, Mr. Kirchhoff was its efficient representative in the United Engineering Society, and on the Building Committee.

In 1910, amid the congratulations of colleagues, employees, and innumerable friends, Mr. Kirchhoff retired from active business as Vice-President and General Manager of a great technical publishing company. Yet he sacrificed of his well-earned leisure yet one more year, to serve the Institute as President through 1911.

The remainder of his life was spent in retirement; but, to those who were privileged to know him most intimately, his long, brave, silent, losing fight against a progressive and fatal disease was a not unworthy close to his useful and distinguished career.

Mr. Kirchhoff was one who secured, because he practiced, loyalty in friendship, justice and kindness in personal intercourse, openness and equity in business. Modest, but not timid; prudent, but progressive; gentle, but firm; full of chivalry and of common sense; indefatigably industrious and inexhaustibly patient, he won victories without making enemies, and crowned with a heroic death the record of a life unstained.

This Board extends to the widow and relatives of Mr. Kirchhoff its profound sympathy in their bereavement, assuring them that the entire membership of the Institute unites in this expression of gratitude for the life and labors, and sorrow in the loss, of an honored colleague and dear friend.

ARIZONA MEETING, SEPTEMBER 18 to 25, 1916

For the first time in its history the Institute will hold a meeting in the State of Arizona.

A few years ago Arizona stood third in the copper-producing districts of the United States. Since that time, with the development of porphyry mines, the output has gone up with great rapidity until it not only is the leading district, but its output at the present time is at the rate of nearly double the Montana output, which now stands second in the list.

In connection with this development, most interesting departures have been made in methods of mining, concentration and smelting. Principles of mining and metallurgy of the greatest interest will be covered by papers which will appear in the *Bulletin* and be presented at Technical Sessions, while ample opportunity will be afforded for those who go to the meeting to see all the points of interest to them. In addition, members will be well rewarded by the marvelous natural features to be seen on the trip, including the petrified forests, the cliff dwellings on the Apache Trail and the Grand Canyon of Arizona.

It is planned to start the special train from New York, but members and guests may join the special at other points en route as indicated by the following schedule:

ITINERARY

Thursday, September 14th

Lv	New York, Grand Central Terminal Dinner in dining car a la carte	5.30 p. m.

New England Members via B. & A. R. R.

Lv	Boston	(Regular train)	2.00 p. m.
Lv	Worcester	(Regular train)	3.12 p. m.
Lv	Springfield Dinner in dining car a la carte	(Regular train)	4.37 p. m.
Ar	Albany	New York Central R. R.	7.45 p. m.
Lv	Albany		9.00 p. m.
Lv	Schenectady		9.32 p. m.
Lv	Utica		11.08 p. m.

Friday, September 15th

Lv	Syracuse	12.30 a. m.
Lv	Cleveland	7.25 a. m.
Lv	Toledo	10.05 a. m.
Ar	Chicago	4.00 p. m.
Lv	Chicago, C. R. I. & P.	8.05 p. m.

Saturday, September 16th

Lv	Kansas City, C. R. I. & P.	11.00 a. m.
Lv	Topeka, C. R. I. & P.	1.05 p. m.
Lv	Hutchinson	6.05 p. m.

Sunday, September 17th

Lv	Santa Rosa, E. P. & S. W. R. R.	6.30 a. m.
Ar	El Paso	2.40 p. m.
Lv	El Paso, Southern Pacific	10.20 p. m.

Monday, September 18th

Ar	Santa Rita	4.00 a. m.

Visit mines of Chino Copper Co. and mill of Empire Zinc Co.

Lv Santa Rita, Santa Fe.............................. 2.00 p. m.
Ar Hurley.. 2.45 p. m.
Visit mill of Chino Copper Co.
Lv Hurley, Sante Fe.................................. 10.15 p. m.

Tuesday, September 19th

Ar Douglas.. 7.00 a. m.
Visit reduction works of Copper Queen Consolidated Mining Co., and of Calumet & Arizona Mining Co.
Technical Session on "Smelting."
Technical Session on "Leaching."
Lv Douglas, E. P. & S. W.......................... 11.00 p. m.

Wednesday, September 20th

Ar Bisbee, Arizona.................................... 1.00 a. m.
Visit to mines of district.
Technical Session, subject "Mining and Geology"
Lv Bisbee, E. P. & S. W............................ 10.00 p. m.

Thursday, September 21st

Ar Globe.. 9.00 a. m.
Visit to mines and reduction works of Old Dominion Copper Co.
Technical Session, subject "Concentration and Flotation."
Technical Session, subject "Fine Grinding."

Friday, September 22nd (Globe)

Visit reduction works of International Smelting & Refining Co., and mills of The Inspiration Consolidated Copper Co., and the Miami Copper Co.

Technical Session, subject "Mining and Smelting."

(The Ray Consolidated Copper Co. will welcome members and guests who desire to visit the mine at Ray. Special arrangements must be made by omitting other features on the program and with the understanding that those who go to Ray must reach Phoenix in time to take the train from there.)

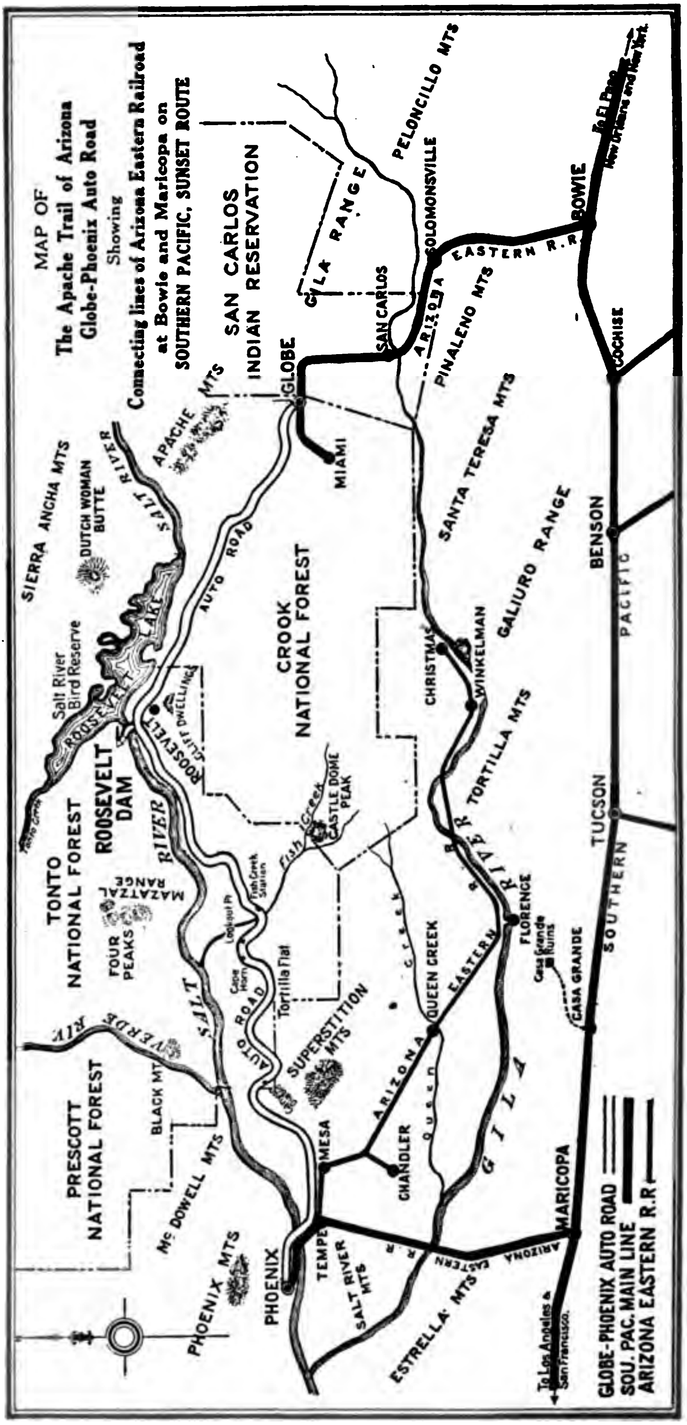
Saturday, September 23rd (Globe)

Lv Globe by automobile over Apache Trail, including visit to Roosevelt Dam.
Ar Phoenix about.................................... 5.30 p. m.
Lv Phoenix, Sante Fe................................ 8.00 p. m.

THE APACHE TRAIL

History records that the ancient trail leading to the ruined cliff dwellings in the Salt River Valley was discovered in 1540 by Francisco Vasquez de Coronado, the Spanish explorer, in his search for the treasures of the "seven cities" of Cibola, the abode of the Zuni Indians—the "Fathers of the Pueblos." Later the trail was used by the Apache Indians and this wild and rugged region was one of their last retreats—the scene of desperate encounters with our troopers in the campaign of 1872. Today the modern motor car follows the same historic and romantic trail through scenery of unusual variety and beauty over the highway between Globe and Phoenix, a distance of 120 miles.

The building of the great Roosevelt Dam, which required the construction of the roadway through the wildest and most rugged sections of the route, has made possible this unique mountain tour, and the consequent transformation of the Salt River and Tonto Creek basins into a great lake region has added a charming aspect to views already grandly impressive.


MAP OF
The Apache Trail of Arizona
Globe-Phoenix Auto Road
Showing
Connecting lines of Arizona Eastern Railroad
at Bowie and Maricopa on
SOUTHERN PACIFIC, SUNSET ROUTE
SAN CARLOS INDIAN RESERVATION
PELONCILLO MTS
GILA RANGE
SOLOMONSVILLE
BOWIE
To El Paso New Orleans and New York
ARIZONA EASTERN R.R.
SAN CARLOS
GLOBE
PINALENO MTS
COCHISE
APACHE MTS
MIAMI
SIERRA ANCHA MTS
DUTCH WOMAN BUTTE
SALT RIVER
AUTO ROAD
SANTA TERESA MTS
GALIURO RANGE
BENSON
CROOK NATIONAL FOREST
Salt River Bird Reserve
ROOSEVELT LAKE
CLIFF DWELLINGS
CHRISTMAS
WINKELMAN
PACIFIC
ROOSEVELT
ROOSEVELT DAM
TONTO NATIONAL FOREST
CASTLE DOME PEAK
Fish Creek
TORTILLA MTS
TUCSON
RIVER
MAZATZAL RANGE
Fish Creek Station
Lookout Pt.
FOUR PEAKS
Cape Horn
Tortilla Flat
GILA RIVER
FLORENCE
SOUTHERN
Casa Grande Ruins
CASA GRANDE
Queen Creek
QUEEN CREEK
SUPERSTITION MTS
VERDE RIV.
PRESCOTT NATIONAL FOREST
BLACK MT
McDOWELL MTS
MESA
CHANDLER
ARIZONA EASTERN
PHOENIX MTS
PHOENIX
TEMPE
SALT RIVER MTS
ESTRELLA MTS
MARICOPA
To Los Angeles & San Francisco
GLOBE-PHOENIX AUTO ROAD
SOU. PAC. MAIN LINE
ARIZONA EASTERN R.R.

CLIFF DWELLINGS.

ROOSEVELT DAM AND SOUTH SPILLWAY.

CLIFF DWELLINGS

At a point four and one-half miles east of Roosevelt, a wagon trail leads up a small canyon to a point where a foot trail may be taken up the steep mountain side to a group of ancient cliff dwellings which are well worth visiting. Another and more extensive group of ruins can be reached two miles farther by trail.

ROOSEVELT DAM

This great dam has a maximum height of 280 ft. When the reservoir is full the water backs up fifteen miles in the Tonto and fifteen miles in Salt River. A hydro-electric plant located below the dam furnished current for use during the construction period and now provides power for irrigation pumping on the Salt River Project and for sale to private consumers.

The Salt River Valley is a land of perennial summer, of oranges, of cotton and of dates. Alfalfa is a staple, and ostrich farming is among the successful enterprises. Irrigation has hastened the valley's progress and confirmed its business stability. Phoenix is its metropolis, an energetic and attractive city, and the capital of the State.

Sunday, September 24th

Ar Grand Canyon	8.00 a. m.
Lv Grand Canyon, Sante Fe	7.40 p. m.

GRAND CANYON OF ARIZONA

The series of tremendous chasms which form the channel of the Colorado River in its course through northern Arizona, reach their culmination in a chaotic gorge, 217 miles long, from 9 to 13 miles wide, and more than 6,600 ft. below the level of the plateau.

Standing upon the brink of the plateau at the point of the canyon's greatest width and depth, the beholder is confronted by a scene whose majesty and beauty are indescribable. It is one of the few widely advertised spots which one need not fear approaching with anticipations too exalted. It has never been adequately described, and never will be.

The poem which follows this itinerary was written by Dr. Raymond in July, 1889, while he was camping with a party of friends near the Grand Canyon. The picture facing the poem, while it fails to show the succession of pillars and "temples" that may be seen from the brink of the Canyon, gives to some extent an idea of the great height and majesty of the walls. The poem and the picture were printed in the souvenir menu of the dinner given to Dr. Raymond by his friends in commemoration of his seventieth birthday.

Monday, September 25th

Lv Albuquerque, Santa Fe	7.00 p. m.

Tuesday, September 26th

Lv La Junta, Sante Fe	7.10 a. m.
Lv Kansas City, Sante Fe	11.00 p. m.

Wednesday, September 27th

Ar Chicago	11.15 a. m.
Lv Chicago, New York Central	12.40 p. m.
Lv Cleveland, New York Central	7.35 p. m.

Thursday, September 28th

Ar Albany	6.27 a. m.
Ar Boston	11.55 p. m.
Ar Grand Central Terminal	9.40 a. m.

Fares quoted below include railroad and sleeping car fares for the entire trip. Fares from New York and Albany also include excess fares applying to Chicago and return owing to extra fast schedule of train.

For the exclusive use of a compartment by one person, one and one-half railroad tickets are required and for a drawing room, two tickets are required.

The arrangements as announced, also railroad and sleeping car fares shown below, are conditional upon one hundred fares New York to El Paso, one hundred twenty-five fares El Paso to Grand Canyon and one hundred fares Grand Canyon to New York.

Starting point	Lower 1 Person	Upper 1 Person	Compartment 2 Persons (Each)	Drawing Room 2 Persons (Each)
New York	$228.50	$217.50	$233.50	$261.00
Albany	216.60	205.60	221.60	249.10
Schenectady	208.96	197.96	213.96	241.46
Utica	205.84	194.84	210.84	238.34
Syracuse	203.84	192.84	208.84	236.34
Rochester	200.60	189.60	205.60	233.10
Buffalo	198.40	187.40	203.40	230.90
Cleveland	188.80	177.80	193.80	221.30
Toledo	186.30	175.30	191.30	218.80
Chicago	176.80	165.80	181.80	209.30
Kansas City	160.30	149.30	165.30	192.80

APPLICATIONS

Those desirous of joining the Tour should promptly communicate with Mr. W. V. Lifsey, Assistant General Passenger Agent, New York Central Lines, 1216 Broadway, New York City. Reservations will be made in the order in which applications are received.

Members of the Arizona Section desiring accommodations on the Special Train should communicate with Mr. Arthur Notman, Box 400, Bisbee, Ariz.

THE GRAND CAÑON OF THE COLORADO.

THE GRAND CAÑON

A thought of God, on earth expressed!
The silence of His perfect rest;
The patience of eternal power;
The ceaseless change from hour to hour;
Forms in alternate gloom and flame
That bide yet evermore the same,
And do but wear such fitful guise,
Reflected in our human eyes,
Which compass only in their range
The things that change or seem to change;
The blended hues of heavenly birth
Beyond the tenderest tints of earth,
That fill and flood her spaces wide
With surges of celestial tide;
The beauty of that awful brink
Where meaner thoughts in rapture sink,
And souls see clear, though eyes grow dim,
While space and time are lost in Him!

Methinks I could not faint or flee
In any conflict yet to be,
Whatever pathway must be trod,
Might I but keep this thought of God!

ROSSITER W. RAYMOND.

July, 1889.

DR. HENRY STURGIS DRINKER

When the suggestion was made by Major General Leonard Wood that the Institute assist in the formation of a National Reserve Corps of Engineers, Dr. Drinker's name occurred first to the Board of Directors as the man who would be most capable of carrying out the proposed plan. Dr. Drinker has long been associated with work in connection with military training and is at present the Chairman of the Governing Committee on Military Training Camps Association which is having such important results at Plattsburg and other places in this country.

As Chairman of the Institute's Committee on National Reserve Corps of Engineers, Dr. Drinker was a member of the Joint Conference Committee of the National Engineering Societies on this subject. He was an earnest worker in the movement which has finally resulted in definite legislation by Congress whereby a Reserve Corps of Engineers is authorized in the new Army Bill which became effective July 1, 1916. Notice of the results of the work was sent to all the members of the National Engineering Societies early in July, and the War Department of the United States has recently issued a circular which has been sent to all the members of the National Engineering Societies, as well as to many other engineers, outlining the line whereby engineers may enroll themselves in the Officers' Reserve Corps. Dr. Drinker has, therefore, fulfilled the mission which he undertook when assuming the Chairmanship of the Institute's Committee. He now asks the permission of the Board of Directors to withdraw as Chairman and has requested that he be replaced in this position by Mr. Arthur S. Dwight, in the belief that the Chairmanship of the Institute's Committee and membership on the Joint Committee should be vested in a man located in New York City, for the better carrying out of the important work now before the Joint Committee. We are happy to say that Dr. Drinker has consented to remain a member of the Institute's Committee, but the completion of the first phase of his work, as Chairman, should not be allowed to pass without an expression of appreciation on behalf of the Institute for his services.

NOMINATIONS FOR OFFICERS

The suggestions of the members of the Institute are very much desired by the Committee on Nominations prior to deciding upon its nominations to fill the places of those officers who retire early in 1917. The Committee must make its final report to the Board of Directors early in October. The members are, therefore, asked to send in promptly their suggestions for Directors as well as for Vice-Presidents and President. The Committee hopes to hear from a large number of the members.

The officers to be elected at the annual meeting in February, 1917, are:

One officer, known as Director and President.
Two officers, known as Director and Vice-President.
Five officers, known as Director.

The attention of members is called to Articles V and VII of the Constitution, and to By-Law XIII, which reads as follows:

"The geographical districts to be considered by the Committee on Nominations shall be as follows, until otherwise ordered by the Board.

District No. 1. New England, New York, and New Jersey, excepting New York

City and district, which is provided for in the Constitution. (N.B.—New York City and district is designated District 0.)

District No. 2. Pennsylvania.
District No. 3. Ohio, Indiana, Illinois, Iowa, and Missouri.
District No. 4. Minnesota, Wisconsin, and Michigan.
District No. 5. Montana, North and South Dakota, Wyoming, Nebraska, Kansas, Washington, Oregon, Idaho, and Alaska.
District No. 6. California and Nevada.
District No. 7. Utah, Colorado, Arizona, and New Mexico.
District No. 8. Louisiana and Texas.
District No. 9. Other Southern States and District of Columbia.
District No. 10. Mexico.
District No. 11. Canada."

An excerpt from Article VII of the Constitution reads as follows:

"In making such selections [namely, the 8 Directors to be elected], the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, so that seven members shall be residents of the district including New York City and the territory within a radius of fifty miles of the headquarters of the Institute, and one member a resident of each of the geographical districts enumerated in the By-Laws."

The officers of the Institute whose terms expire are as follows:

President, L. D. Ricketts (not eligible for re-election).

Past President, Benjamin B. Thayer.

Vice-Presidents, H. C. Hoover, District 6; Joseph W. Richards, District 2.

Directors, Reginald W. Brock, District 11; Albert R. Ledoux, District 0; D. C. Jackling, District 7; Charles W. Merrill, District 6; Henry L. Smyth, District 1.

Last year an unusually large vote was cast for officers, and it is hoped that members will show a similar interest and activity in communicating to the Committee their views regarding the men best fitted to fill the vacancies.

Communications should be sent to the Chairman of the Committee on Nominations, 1208 Hollingsworth Bldg., Los Angeles, Cal.

SEELEY W. MUDD,
Chairman, Committee on Nominations.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JULY 25, 1916

A special meeting of the Board of Directors was held on July 25, 1916, pursuant to notice duly given according to the By-Laws.

This special meeting was for the purpose of giving authority for the signing of a contract with the American Society of Civil Engineers, as mentioned in another place in this Bulletin. The interest in the matter is indicated by the attendance of 11 members of the Board, some of whom came from a long distance.

An invitation was received from the St. Louis Section of the Institute and from members in and near St. Louis to hold the 115th Meeting of the Institute in and near St. Louis in late September or early October, 1917. This invitation was accepted.

Announcement was made that the Mexican Institute of Mining and Metallurgy had been dissolved and reorganized as the Mexican Section of the Institute.

The sum of $250 was appropriated to the Nevada Section.

The usual election of members took place.

Resolutions were passed regarding the death of Charles Kirchhoff.

AMERICAN SOCIETY OF CIVIL ENGINEERS

The American Society of Civil Engineers, by a vote of its membership of six to one, has favored the acceptance of the invitation of the three present Founder Societies, namely, the American Society of Mechanical Engineers, American Institute of Electrical Engineers and American Institute of Mining Engineers, to become a fourth Founder Society and share with the rest the occupancy of the Engineering Societies' Building.

Fortified by this vote, the Board of Direction of the American Society of Civil Engineers has caused a contract to be entered into between that Society and the present Founder Societies and the United Engineering Society.

Work has already begun on adding three stories to the Engineering Societies' Building at a cost not to exceed $250,000. This cost will be borne by the American Society of Civil Engineers, thereby making its contribution approximately equal to that of the other Founder Societies.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period July 10, 1916 to August 1, 1916:

W. A. Barnes, San Francisco, Cal.
W. L. Bell, Davenport, Fla.
James P. Clarke, Jr., Little Rock, Ark.
R. A. Carhart, Mitchell, S. D.
G. M. Colvocoresses, Humboldt, Ariz.
W. E. Defty, Phoenix, Ariz.
Carl F. Dietz, Worcester, Mass.
W. J. Mead, Madison, Wis.
Charles Mentzel, San Pedro, N. M.
H. H. Miller, New York, N. Y.
W. E. Pomeroy, El Paso, Tex.
M. K. Rodgers, Los Angeles, Cal.
S. C. Rodgers, Guanajuato, Mex.

John M. Baker and Hamilton W. Baker have resigned as general manager and mine superintendent respectively of the Baker Mines Co., of Oregon. After Aug. 1, their address will be 43 Emerson St., Denver, Colo.

Noel Cunningham has opened an office at 200 Fifth Avenue, New York, where he will continue his work as consulting metallurgical engineer.

George E. Farish, of New York, has moved his western office from Denver, Colo., to the Nevada Bank Building, 14 Montgomery St., San Francisco, Cal.

Captain J. P. Hodgson, formerly mine superintendent of the Copper Queen, is now acting in a consulting capacity in the mining department of Phelps, Dodge & Co.

William F. Jahn, formerly mill superintendent for the New York and Honduras Rosario Mining Co. at San Juancito Honduras, has accepted a position as mill superintendent for the Tough-Oakes Gold Mines, Limited at Kirkland Lake, Ontario, Canada.

Barnard MacDonald has moved his office from Los Angeles to Mills Bldg., El Paso, Tex.

H. K. Najarian has accepted a position with Bradley Bruff & Lakarthe of San Francisco, and is employed on designing work at the new smelter of the Missouri Cobalt Co. at Fredericktown, Mo.

Fred S. Porter has accepted a position with the Canadian Klondike Mining Co., Dawson, Yukon Territory.

R. W. Schultz, formerly with the Mond Nickel Co., has become associated with the Minerals Separation, Ltd.

Gerald Sherman is now general superintendent of the mine department of the Copper Queen.

J. Frank Sinclair, general foreman of the lower divisions of the Copper Queen, has been promoted to the position of superintendent of those divisions.

F. W. Traphagen has resigned as professor of metallurgy at the Colorado School of Mines and is now president of the Colorado Metal Mining & Reduction Co.

Felix E. Wormser is now with the Snake River Mining Co., Huntington, Ore.

Pope Yeatman, Mining Engineer, of 120 Broadway, New York, N. Y. will resume independent practice on Sept. 1, 1916.

POSITIONS VACANT

Engineer to take charge of construction of smelter and mill. Experience with blast furnaces, roasters, belt-conveying systems and general machinery, essential. Salary $250 per month up according to experience. No. 129.

Mine Engineer.—Man with experience, ability and initiative wanted to handle an organization, direct sampling, surveying, mapping and record keeping. Experience in laying out development work, haulageways, shaft pockets and ore-handling systems, particularly desired. No. 130.

Recent technical graduate for experimental work with a metal refining company. Laboratory work first to acquaint worker with methods, then research work. Good opportunity for advancement. Beginning salary $100 a month. No. 132.

Mine superintendent. Man of experience and energy wanted in mine being developed to produce 5,000 tons of iron-ore per day from open-cut and underground. About to start underground operations. Expect to follow methods of low-grade copper mines of Southwest. No. 133.

Two fellowships for metallurgical research have been established by the University of Idaho. Each fellowship carries $500 per annum. No. 134.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, Cornell M. E., aged 31, married. Will soon complete 2 years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15, in the United States. No. 299.

Member, technical graduate, who has specialized in the treatment of gold and silver ores, desires position as mill superintendent or foreman; 7 years' practical experience. Highest references as to character and ability to get results. No. 307.

Member, graduate mining engineer. Married. Age, 38. For past 6 years, general superintendent of large silver mine in Mexico. Desires connection with responsible mining company, preferably in the United States. Has had 15 years' experience in the United States and Mexico, including mine engineering, sampling, examinations, milling and mine superintendence. No. 312.

Member, specialist in the mining and reduction of quicksilver ores, open for engagement Sept. 1. 19 years' experience. 14 years as Superintendent and Manager. Aged 39, married. No. 313.

Metallurgist, 4 years' experience in electric furnace operation on steel, ferro-alloys and smelting of ores, both practical and research, desires position as metallurgist or superintendent. No. 314.

Member, aged 44 years, superintendent of coal mines for 13 years and of coke ovens for 4 years, desires to locate in altitude of less than 3,000 ft. No. 315.

Member, technical mining graduate. 5 years' experience in coal and metal mines and construction work. Married. Aged 27. Employed at present. Interview in Salt Lake City or Denver. No. 316.

Member, technical graduate, 15 years' experience in engineering and mining in Western States, desires position as manager or superintendent. Age, 38. Married. Excellent references. No. 317.

Member, American, aged 39, married, desires position as superintendent or mine superintendent. 14 years' experience in opencut and underground work. 12 years' Spanish, Portugese, Cuban and Mexican labor. Fluent Spanish. Good training in first aid and safety first. Specialty of high explosives and steam-shovel work. Good costs. Go anywhere, but tropics preferred. References. Interview—New York or Philadelphia. No. 318.

LOCAL SECTION NEWS

MONTANA SECTION

J. L. BRUCE, *Chairman*

W. C. SIDERFIN; *Vice-Chairman*

WALTER E. GABY, *Secretary-Treasurer*, 834 W. Granite St., Butte, Mont.

N. B. BRALY W. T. BURNS

A special meeting of the Directors of the Montana Section was held on July 24, 1916, with W. C. Siderfin, Vice-Chairman, presiding.

The meeting was called for the purpose of electing a Secretary-Treasurer to fill the vacancy created by the resignation of M. H. Gidel.

Upon proper motions, the resignation of M. H. Gidel was accepted, and Walter E. Gaby was elected to fill the office for the balance of the fiscal year.

No further business was transacted, and upon motion the meeting adjourned.

WALTER E. GABY, *Secretary*.

SOUTHERN CALIFORNIA SECTION

C. COLCOCK JONES, *Chairman*

ALVIN B. CARPENTER, *Vice-Chairman*

F. J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Building, Los Angeles, Cal.

A. B. W. HODGES E. A. MONTGOMERY

R. A. PEREZ W. F. STAUNTON

The Annual Meeting of 1916 was held on June 28, at 7:30 p. m. There were present the Messrs. A. B. Carpenter, A. B. W. Hodges, F. J. H. Merrill, Seeley W. Mudd, R. A. Perez, and A. J. Underwood. The Chairman, Mr. Mudd, presided.

The Secretary read the minutes of the annual meeting of 1915, and of the meeting of September 28, 1915, as well as of the last meeting of the Executive Committee of May 25, 1916. On motion these minutes were ordered approved as read.

The Secretary-Treasurer then read his reports for the past year, and it was voted that these reports be approved and entered in the minute book.

A vote of thanks was given to the Chairman and the Secretary-Treasurer for their work during the year just past.

The amendment to Article III of the By-Laws, announced in the notice of the Annual Meeting, was put to vote and unanimously carried.

The Secretary read the minutes of the meeting of the Nominating Committee on May 31, and announced the result of the annual election as indicated above.

After discussion of plans for attending the Arizona meeting, the meeting adjourned.

F. J. H. MERRILL, *Secretary*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

COAL, ITS ECONOMICAL AND SMOKELESS COMBUSTION. By J. F. Cosgrove. Philadelphia, 1916.

COKE OVEN ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEAR 1915. Technical Paper 151, U. S. Bureau of Mines, Washington, 1916.

MINE SAMPLING AND VALUING. By C. S. Herzig. San Francisco, 1914.

TECHNOLOGY OF MARBLE QUARRYING. Bulletin 106, U. S. Bureau of Mines, Washington, 1916.

Chemistry and Microscopy

HANDBÜCH DER PHYSIKALISCH CHEMISCHEN TECHNIK FÜR FORSCHER UND TECHNIKER. By Kurt Arndt. Stuttgart, 1915.

MICROSCOPICAL DETERMINATION OF THE OPAQUE MINERALS. By Joseph Murdoch. New York, J. Wiley & Sons, 1916. (Gift of publisher.) Price $2.00.

[NOTE.—Treats of the technique of mineral examination by treatment with various reagents under the microscope. Ingeniously arranged tables are provided. W. P. C.]

Mineralogy and Mineral Resources

CHILE. 1915. (Gift of Eduardo Carrasco.)

LIMESTONES AND MARLS OF THE COASTAL PLAIN OF GEORGIA. A report. Bulletin 21, Georgia Geological Survey. Atlanta, 1916.

ZIRCON, MONAZITE AND OTHER MINERALS USED IN THE PRODUCTION OF CHEMICAL COMPOUNDS EMPLOYED IN THE MANUFACTURE OF LIGHTING APPARATUS. Bulletin 25, North Carolina Geological and Economic Survey. Raleigh, 1916.

Power

BALANCING OF ENGINES, STEAM, GAS AND PETROL. By Archibald Sharp. New York, 1907.

POWER TRANSMISSION BY LEATHER BELTING. By R. T. Kent. New York, J. Wiley & Sons, 1916. (Gift of Publisher.) Price, $1.25 net.

[NOTE.—Belting practice has changed greatly in the past fifteen years. The literature is buried in the transactions of engineering societies and in technical journals. This book seeks to gather this into a compilation of the greatest service to the user of belting. W. P. C.]

General

AIRCRAFT IN WARFARE. By F. W. Lanchester. New York, 1916.

THE AUTHENTIC HISTORY OF THE UNITED STATES STEEL CORPORATION. By Arundel Cotter. New York, Moody Magazine & Book Co., 1916. (Gift of publisher.)

[NOTE.—The author admits a prejudice in favor of the corporation. It is an interesting contribution to industrial history. W. P. C.]

BIOGRAPHY OF AUGUSTE DAUBRÉE. June 25, 1814–May 29, 1896. (Gift of Miss F. Daubrée.)

BUSINESS AFTER THE WAR. By G. E. Roberts. New York, 1916. (Gift of National City Bank.)

ENGINEER'S YEAR BOOK, 1916. By H. R. Kempe. London, 1916.

THE LEGACY OF THE EXPOSITION. INTERPRETATION OF THE INTELLECTUAL AND MORAL HERITAGE LEFT TO MANKIND BY THE WORLD CELEBRATION AT SAN FRANCISCO, 1915. San Francisco, 1916. (Gift of Panama Pacific International Exposition Co.)

Company Report

TEMISKAMING AND NORTHERN ONTARIO RAILWAY, MINING INDUSTRY IN THAT PART OF NORTHERN ONTARIO SERVED BY. 1915. Toronto, 1916. (Gift of Temiskaming and Northern Ontario Railway.)

Trade Catalogs

THE ALLEN CONE CO., El Paso, Texas. Description of Allen Cone for classifying, tube milling, etc.

CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bulletin E-41. Duntley electric tools for street and interurban railways. 1916.

CHICAGO STEEL TAPE CO., Chicago, Ill. Price list. 1916.

INGERSOLL RAND CO., New York, N. Y. Form 9024. Steam condensing plants. June, 1916.

NATIONAL TRANSIT PUMP AND MACHINE CO., Oil City, Pa. Bulletin 502. Horizontal oil engines.

WIEDEKE, GUSTAV & CO., Dayton, O. Catalog No. 48. Ideal tube expander and cutter. 1915.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period July 10, 1916, to July 31, 1916:

Boyd, Robert Russell.........Supt., 'Arizona Commercial Min. Co., Globe, Ariz.
Bradley, Henry Franklin, Chem., American Smelt. & Ref. Co., Murray, Utah.
Brown, Thomas Ellis, Jr......................55 Hill St., Morristown, N. J
Cary, Webster P., Expert Operator, General Engineering Co., Salt Lake City, Utah.
Clark, Alonzo Webster................2222 Scottwood Ave., Toledo, Ohio.
Corp, Clifford...........Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
Crombie, W. Ransom, Min. Engr......Detroit Copper Min. Co., Morenci, Ariz.
Delaney, Eugene A., Chief Engr., Berwind-White Coal Mining Co., Windber, Pa.
Elbelt, William Henry, Dept. Foreman., Chino Copper Co., Hurley, N. Mex.
Foster, L. Julio, Mgr., Minas i Fundicion "La Junta," San Bernardo, Chile, South America.
Gaskill, James Phillips, Genl. Mgr., Ajo Cons. Copper Co., Ajo, Ariz., via Gila, Ariz.
Hays, Fred N., Asst. Steam Engr., Carnegie Steel Co., Farrell & Sharon Works, Sharon, Pa.
Hood, Karl Kedzie, Min. Engr.............American Zinc Co., Mascot, Tenn.
Johnson, Ernest C., Prest. & Treas., Johnson Engineering Works, First National Bank Bldg., Chicago, Ill.
Kniffin, Lloyd Malcolm, Asst. Supt., United States Metals Ref. Co., East Chicago, Ind.
Knowles, Benjamin Wallis, Min. Engr., Hedley Gold Mining Co., Hedley, B. C., Canada.
Labbe, Armand Louis........Chem., American Smelt. & Ref. Co., Murray, Utah.
Lang, Sidney A., Min. Engr., Braden Copper Co., Rancagua, Chile, South America.
Lees, John W.................Genl. Supt., Inland Steel Co., Indiana Harbor, Ind.
Leopold, F. N.........................40 North Dearborn St., Chicago, Ill.
McAfee, Dan Shields, The Dorr Company, 17 Battery Pl., New York, N. Y.
McClain, Henry G., Genl. Supt., Liberty Bell Gold Min. Co., Telluride, Colo.
Marshall, Albert E., Chemical Engr..........Box 75, Sparkill, New York, N. Y.
Milligan, William Edgar, Chem., Braden Copper Co., Rancagua, Chile, South America.
Moses, Frederick Gallaway, Flotation Engr., General Engineering Co., 159 Pierpont St., Salt Lake City, Utah.
Newby, Jerry B., Oil Geol................Gypsy Oil Co., Box B, Tulsa, Okla.
Palmer, Edgar, Pres., New Jersey Zinc Co., 55 Wall St., New York, N. Y.
Pattinson, Reginald Lancaster, Producer of Natural Gas and Petroleum, 29 5th St., Chatham, Ont., Canada.
Rathbun, R. B., Asst. Research Engr., American Smelt. & Ref. Co., Murray, Utah.
Rigby, William Arthur, Chief Engr., Witherbee, Sherman & Co., Mineville, Essex Co., N. Y.
Sargent, Arthur E., Met. Engr............................Terry, So. Dak.
Schimerka, Francis S., Met. Chemist.......Shannon Copper Co., Clifton, Ariz
Stowell, George Elwin, Instructor in Mining, Oregon Agricultural College, Mining Dept., Corvallis, Ore.
Street, Lloyd G., Met. Engr...........Mineral Point Zinc Co., Depue, Ill.
Thayer, Warren N., Prof. of Geology, Ohio Mechanics Institute, Cincinnati, Ohio.
Thompson, David P., Asst. Genl. Supt., Inland Steel Co., Indiana Harbor, Ind.
Titzell, George Graham, Jr....................110 So. First St., Duquesne, Pa.
Velasco, Louis Ibanez, Engr......Lehigh Coal and Navigation Co., Lansford, Pa.
Ward, Howard Ridgely, Min. Engr., American International Corpn., 120 Broadway, New York, N. Y.
Wardell, John Wilfrid, Mech. and Const. Engr., Spassky Copper Mine, Ltd., Spassky Zavod, Karagandy, Akmolinsk Dist., Siberia.
Williams, Michael Thomas, Genl. Mgr., Edna May Gold Min. Co., N. L., Westonia, Western Australia.

Junior Members

AMBLER, HARRY A., Student..................4050 Russell Ave., St. Louis, Mo.
ANDERSON, BARCLAY GLADSTONE, Asst. in Geol. Dept., Burro Mountain Copper Co., Tyrone, N. Mex.
BICKNELL, HAROLD LEWIS, Furnace Operator, Snyder Electric Furnace Co., 324 N. Grove Ave., Oak Park, Ill.
DIEFENDERFER, A. J......................34 North Oakland Ave., Sharon, Pa.
FINKELDEY, WILLIAM H., Min. Engr., Compania Minera y de Fomento Watkins, Fomento, Prov. Santa Clara, Cuba.
LEVY, MILTON MORTIMER.................1 Vernon Apts., Salt Lake City, Utah.
MILLER, ROY HARRISON, Milling....Anaconda Copper Min. Co., Anaconda, Mont.
MILYKO, ALEXANDER, Asst. Geol., Producers Oil Co., Box 75, Houston, Tex.
PETZEL, CHARLES L., Student............Columbia University, New York, N. Y.
WALTER, ALPHONS R., Learner of concentrating plant, Bethlehem Steel Co., So. Bethlehem, Pa.
WORMSER, FELIX E......P. O. Box 156, Snake River Mines Co., Huntington, Ore.

Total Membership, July 31, 1916..............................5,654

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period July 10, 1916 to July 31, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Hans Claudius Anchor, So. Porcupine, Ont., Canada.

Proposed by H. P. De Pencier, J. Ralph Scott, C. E. Rodgers.

Born 1857, Hasle, Bornholm, Denmark. Attended Public School in Denmark up till 1871. Studied under my Father at Coal Mine (Lignite), Island of Bornholm, till 1874. 1875–76, Mining and railroad construction, Comstock, Nev. Learned assaying for the common metals, 1877. 1877–85, Worked in different mines through the State of Nevada. 1885–86, Black Jack, Silver City, Ida. 1887–97, De Lamar, Ida. 1897–98, Twin Devils, Ida. (copper). 1898–1902, Boise Basin, Ida. (gold). 1902–05, Bully Hill, Cal. (copper). 1906–07, Hermoza Copper Co., N. Mex. 1908–10, Traveling, mostly examining prospects. 1910, Sampled and outlined orebody Dome Mines, So. Porcupine.

Present position: Charge of Developments, Dome Extension Mines Co., Ltd.

Edward Otto Forstér Brown, London, E. C., England.

Proposed by M. C. Scheble, Edgar Rickard, H. Foster Bain.

Born 1881, Cardiff, Great Britain. 1895–1900, Winchester College. 1900–05, Training mainly at collieries. 1905, Obtained first class Mine Managers' Certificate. 1905–07, Inspecting collieries and mines, S. Africa, China, Japan, United States, Canada. 1908–12, Cons. Engr., 7 Cohring Cos. in Coahuila, Mex., and reporting in coal fields in British Columbia and Pennsylvania.

Present position: Managing and Cons. Engr., for a number of coal mines in Kent, S. Wales, N. Wales and Wolpam Mines in Spain.

Gilbert Haven Cady, Urbana, Ill.
Proposed by J. N. Ede, Thomas T. Read, William H. Shearman.
Born 1882, Chicago, Ill. 1905, Northwestern Univ., A. B. 1907–08, Graduate student in Geology, Northwestern Univ. 1909–10, Graduate student in Geology, Yale Univ. 1911–12, Graduate student in Geology, Illinois Univ. 1912–13, Graduate student in Geology, Chicago Univ. 1907–16, Member of the staff of the Illinois Geol. Survey with rank of Geologist since 1914. Have been in continuous employment on the survey since 1914. Have especial charge of the coal investigations and have prepared two published reports on Illinois Coal Districts—Longwell District and Jefferson, Franklin and Williamson Counties District—and have other reports in preparation. Have also presented for publication manuscripts for two U. S. Geol. Survey folios concerning West Frankfort-Galatia quadrangles and Hennepen La Salle quadrangles. 1913–14, Instructor in Geology, Northwestern Univ. Am a member of the U. S. Geol. Survey with rank as Asst. Geol.
Present position: Geologist, Illinois Geol. Survey.

H. T. Chen, Yunnan Prov., Yunnan, China.
Proposed by Geo. O. Scarfe, Robert H. Bedford, A. D. Foote.
Born 1878, Hunnan Prov., China. Attended Schools in China, and Japan. Berkeley High School. 1911, Univ. of Cal., College of Mining, B. S. 1911, winter, 1912, spring, was the director of Industrial Dept. of Hunnan Prov. 1912, summer, engaged to be Asst. Engr., Yunnan Tin Trading Co. 1913, spring, was promoted to be Engr. in Chief of Yunnan Tin Trading Co. The works are entirely modern, containing 52 washing tables and 6 reverberatory furnaces.
Present position: Engr. in Chief, Yunnan Tin Co.

Charles L. Constant, New York, N. Y.
Proposed by H. W. Hardinge, Ronald Clark, R. B. T. Kiliani.
Born 1856, Cincinnati, O. 1877, Grad. from School of Mines, Columbia, as Civ. and Min. Engr.

James Belcher Etherington, Winthrop, Mass.
Proposed by D. A. Lyon, Oliver C. Ralston, C. A. Wright.
Born 1851, Nova Scotia. 1865, Mixed Schools in Nova Scotia. 1872, Miss Boynton's Private School, Lynn, Mass. 1873, Bryant and Stratton Commercial School, Boston. 1874–79, Bookkeeper for Oscar F. Howe, Woodenware, Boston, Mass. 1881–92, Dealer, Mfgr. & Exporter, Manufactures of Wood, firm of J. B. Etherington & Co., Boston. 1892–98, Mfgr. of hardwood lumber, Bradford, Pa., firm of Bradford Hardwood Lumber Co. Since 1901, have been active principally in magnetic separation of ferruginous ores. Knowledge of the subject gained chiefly through study of Encyclopedia Brittanica and through association with Dr. Campbell, Dr. S. P. Sharples, Boston, and various others in the zinc business in the United States, Canada and Belgium.
Present position: Zinc Concentrating Co., Boston, Mass.

Julio F. Fernandez, Manzanillo, Cuba.
Proposed by Louis F. Chibas, E. Chibas, Thomas T. Read.
Born 1882, Havana. 1906–10, Asst. Engr., Public Works Dept. 1910–12, Chief Engr. of Manzanillo Water and Light Co. I made the project of Manzanillo Water Work. Have been the Concessionary. Studied in Spain, Zaragozo University. Afterward in Cuba by correspondence with American School of Correspondence of Chicago, Ill. 1912–14, I made the Manzanillo project of city pavement, 20 miles of concrete, and at the same time the studies of Manzanillo and Wignero R. R., 52 miles. I am the Promoter and the Director. 1914–16, Promoter and studies of Manzanillo Electrical Traction Co. In this moment I am exploiting two copper mines in Manzanillo and am Promoter of the Manzanillo Mines Copper Co.
Present position: Director and Promoter of all foregoing work.

Rollins Sanders Foster, Butte, Mont.
Proposed by C. W. Goodale, F. A. Linforth, B. B. Thayer.
Born 1885, Montana. 1900–03, Montana State School of Mines. 1904, Univ. of Cal. 1905–07, Miner, sampler and timekeeper, Gagnon Mine for C. G. Adami. 1907–11, Asst. Surveyor and Shift boss, North Butte Min. Co. 1911–13, Prospector, Belgian Congo, Société Forestière et Minière du Congo. 1914–15, Miner and Shift boss, Anaconda Copper Min. Co. 1915–16, Asst. Safety Engr. and Safety Engr., Anaconda Copper Min. Co.
Present position: Safety Engr., Anaconda Copper Min. Co.

G. V. Foutz, Minas de Ponupo, La Maya, Oriente, Cuba.
Proposed by Charles F. Rand, E. Aguilera, Pedro Aguilera.
Born 1893, Mason-Dixon, Pa. 1911, Waynesboro, Pa., High School. 1915, Pennsylvania State College, B. S., Industrial Chemistry. 1915, Aetna Explosives Co., Emporium, Pa., Carnegie Steel Co., Duquesne, Pa., Aguilera & Co., Santiago de Cuba.
Present position: Chemist.

Walter Irving Garms, Hayden, Ariz.
Proposed by Louis S. Cates, D. D. Moffat, William T. MacDonald.
Born 1886, San Franciso, Cal. 1900–05, Technical course in Cal. School of Mech. Arts, San Francisco. 1905–10, Univ. of Cal., B. S. in College of Mining. 1907, Mining, South Eureka Min. & Mill. Co., Sutter Creek, Cal. 1910, Sampling and Geology, Ray Cons. Copper Co., Ray, Ariz. 1911–16, Milling and construction, Ray Cons. Copper Co., Hayden, Ariz.
Present position: General Mill Foreman, Ray Cons. Copper Co.

George William Hain, Lansford, Pa.
Proposed by Edwin Ludlow, W. G. Whildin, J. B. Warriner.
Born 1886, Tremont, Pa. 1909, Lehigh Univ., E. M. 1909–12, Min. Engr., Juragua Iron Co., Santiago de Cuba. 1912–15, Mgr., West Virginia Engrg. Co., Fairmont, W. Va.
Present position: 1915 to date, Asst. to Dist. Supt., Lehigh Coal and Navigation Co.

Daniel R. Hull, Kenosha, Wis.
Proposed by Milo W. Krejci, John H. Klepinger, E. S. Bardwell.
Born 1888, Southington, Conn. 1908–11, Sheffield Scientific School, Yale Univ. 1911–16, Met. Chemist, Kenosha Branch, American Brass Co.
Present position: American Brass Co.

Arthur D. Knowlton, Salt Lake City, Utah.
Proposed by David Lemmon, L. O. Howard, J. H. Winwood.
Born 1881, Tooele Co., Utah. 1903, Univ. of Utah, B. S. in Min. Engrg. 1903–05, with State Engineer, Utah. 1905–07, with City Engineer, Salt Lake City. 1907–16, Min. Engrg., U. S. Mineral Surveyor, Utah and Nevada. 1916, Cons. Engr., Boston Development Co.
Present position: Engr., Boston Development Co.

Ray P. McGrath, San Francisco, Cal.
Proposed by Frederick K. Copeland, Stillman Batchellor, Edward Jüssen.
Born 1883, Stratford, N. H. 1906, Dartmouth College. 1906, worked for Sullivan Machinery Co. Have been with this company since that time acting as a sales engineer, testing, installing and selling mining machinery.
Present position: Dist. Mgr., Sullivan Mach. Co.

Will E. McKee, Warren, Ariz.
Proposed by John C. Greenway, Ira B. Joralemon, W. B. Gohring.
Born Teppreanor Co., Ind., 1866. 1890, Univ. of Ill., B. S. in Mech. Engrg. 1890–91, Draughtsman, Link Belt Mach. Co. 1891–92, A. L. Ide & Son, Springfield, Ill. 1892–93, Construction, Pabst Brewing Co. 1894, Engr., Sanger Bros., Dallas, Tex. 1895–96, Cons. Engr., E. F. Osborn, Chicago. 1897–1905, Master Mechanic, Cleveland Cliffs Iron Co., Ishpeming, Mich.
Present position: 1905 to date, Supt. Machinery, Calumet & Arizona Mining Co.

John Henry Miles, Trinity Center, Cal.
Proposed by Benjamin F. Tibby, Harry L. Mead, Frank E. Johnson.
Born 1882, Calumet, Mich. Butte Grammar School. Four years, Butte Business College. 1905–07, Supt. construction, Boston Machine Shop Co., Oroville, Cal. 1907–11, Master mechanic, Yuba Cons. Gold Fields Dredging Co., Hammonton, Cal. 1911–13, Supt. of Dredge Folsom Division, Natomas Cons. of Cal. 1913–16, Genl. Supt., Boston and Idaho Gold Dredging Co., Idaho City, Ida.
Present position: Genl. Supt., Alta Bert Gold Dredging Co.

Henmann Wilhelm Paul, Yokohama, Japan.
Proposed by Walter A. Schmidt, L. D. Ricketts, Thomas T. Read.
Born 1878, Weddingen, Germany. 1904, Engr., Freiberg in Saxony. 1905–06, Borhumer Verein für Bergben und Gusstahlfabrikation, Borhum in Westphalia, as Chemist. 1906–09, technical expert for their Mining, Metallurgy and Ore Business,

Otto Reimers & Co., Yokohama, Japan. 1910, Metallgesellschaft, Frankfort and Telge and Schroeter, Shanghai, China, for their Mining and Ore Business in China. 1911–12, Otto Reimers & Co., Yokohama, Japan.

Present position: Min. and Met. Engr. Representation of the Metallgesellschaft and the Metallbank, Metallurgische gesellschaft, Frankfort, for Japan, and of the International Precipitation Co., Los Angeles, Cal., for Japan.

Francis Stuyvesant Peabody, Chicago, Ill.

Proposed by Charles H. MacDowell, Robert W. Hunt, Carl Scholz.

Born 1859, Chicago, Ill. 1881, Grad., Sheffield Scientific School, Ph. B. Mining coal 20 years.

Present position: Pres., Peabody Coal Co.

H. J. D. Penhale, Oruro, Bolivia, S. Amer.

Proposed by M. G. F. Söhnlein, P. F. Bliek, H. L. Venables.

Born 1893, Illogan, Cornwall, England. Cornwall County Schools. School of Metaliferous Mining, Redruth, Cornwall. 1911, Asst. Surveyor, South Crofty Mines, Cornwall. 1912, Engr. to the Devon United Mines, Ltd. 1913, Mgr. (Mine) Antequera Mines, Bolivia. 1913–14, Surveyor, Compania Minera, Nane, Bolivia. 1914–15, Asst. Mine Mgr. and Surveyor, Cia. de Minas de Cobre de Gatico.

Present position: Engr., House of Penny y Duncan.

Kiva Pommerantz, Santiago, Chile, S. Amer.

Proposed by Louis A. Wright, Mark R. Lamb, F. H. Barclay.

Born 1884, Kuhurlui, Besarabia, Russia. 1901, Grad., B. S., in Komrat, Russia. 1901–05, Mining Academy, Freiberg, Saxony, "Diplom-Ingenieur". 1905–08, Surveyor and later on Superintendent of one of the zinc mines belonging to the "Minengesellschaft Fr. Speidel, Tharsos" Turkey. 1908–11, Supt., Manganese and Copper Mines of Hopa (Asia Minor). 1911–14, Technical Mgr., "Société Hongroise d'Etudes de l'Industrie Minière et Metallurgique," Budapest.

Present position: 1914 to date, Cons. Engr., Ore Trading Co., Ltd.

Harold Carmichael Robson, Akmolinsk, Siberia.

Proposed by Walter G. Perkins, C. G. Hall, Herbert C. Woolmer.

Born 1890, Penzance, Cornwall, England. 1897–1900, The Abbey School, Penzance, Cornwall, England. 1900–05, St. Mary's School, Penzance. 1905–09, School of Mines, Penzance, receiving school diplomas. 1909–10, Practical work, Dolcoath Mine, Camborne, Cornwall, England, Min. & Mill. 1910–11, Asst. to W. Roberts, Siberian Syndicate, London Wall, E. C., England, who reported on various concessions in the Altai District, Siberia. 1911–14, Junior Asst. Engr., Spassky Copper Mine, Ltd.

Present position: 1914 to date, Asst. Engr. in Smelter, Spassky Copper Mine, Ltd.

Leon Fair Russ, Tampico, Mexico.

Proposed by C. W. Hamilton, E. DeGolyer, Lewis C. Chapman.

Born 1886, Coushetti, La. 1909, Univ. of Texas, B. A. 1909–10, special work geology and chemistry, Univ. of Texas. 1907–10, Geologic field work, Field Asst., U. S. Geol. Survey. 1911, Geol., Cia. Mex. de Pet. "El Aguila," S. A.

Present position: Geol., Cia. Mex. de Pet. "El Aguila," S. A.

Jesse D. Smith, Galena, Ill.

Proposed by Arthur Thacher, H. A. Wheeler, J. F. Thompson.

Born 1860, New Madrid, Mo. Few years in Public Schools. 1897–1905, Central Lead Co., Flat River, Mo. 1905, Federal Lead Co., Flat River, Mo. 1906, St. Louis Smelting & Refining Co., Flat River, Mo. 1907–1916, Mineral Point Zinc Co., Mineral Point, Wis.

Present position: Supt., Black Jack Mine.

Thomas B. Sturges, Pittsburgh, Pa.

Proposed by Richard T. Dana, Clarence B. Sturges, A. H. Carnahan.

Born 1884, Wilkes-Barre, Pa. 1901, C. P. Course, Wilkes-Barre, Pa., High School. Other geological and engineering knowledge gained by private study and field work. 1902, Rodman and inspector, Webster Coal & Coke Co., Cresson, Pa. 1903–06, Min. Engr., Cons. & Fernwood Collieries, Hillside Coal & Iron Co., Dunmore, Pa. 1906–16, Genl. Mgr., Pennsylvania Drilling Co. Prospecting coal and other mineral land in United States & Canada, Cuba, Central America, etc.

Present position: Vice-Pres. & Genl. Mgr., Penn. Drilling Co.

Haruyoshi Tomita, Akitaken, Japan.
Proposed by E. P. Mathewson, H. W. Aldrich, Ichiro Kamimura.
Born 1887, Kumamoto, Japan. 1906–09, The 5th Higher School, Kumamoto. 1909–12, Tokyo Imperial Univ.
Present position: 1912 to date; Met. Engr., Osarusawa Mine, Mitsubishi Co.

Frederick H. Vahrenkamp, Salt Lake City, Utah.
Proposed by David Lemmon, G. B. Wilson, L. O. Howard.
Born 1870, Concordia, Mo. 1888, High School. 1892–96, Private studies in engineering and geology. 1898–1904, Business associate of Prof. Mitchell of Mitchell & Vahrenkamp, Los Angeles, Cal. 1905–06, Prest., Anvil Springs Gold Mining Co. 1906–08, Prest., Bonnie Clare Mining Co., Goldfield, Nev. 1908–09, E. M. Humboldt Exploration Co., New York, N. Y. 1909–12, Genl. Mgr., Coates Motor Car Co., New York, N. Y. 1913–14, Prest., Onyx Travertine Co., Salt Lake City, Utah.
Present position: Prest., Boston Development Co., Boston Nevada Copper Co.

William Wilke, Buffalo, N. Y.
Proposed by C. W. Stimpson, R. F. Barker, W. D. Leonard.
Born 1855, Darmstadt, Germany. Public and High School in Darmstadt, Germany. 1872–77, Polytechnicum Darmstadt, M. E. 1881–86, Asst. Engr., Midvale Steel Co., Philadelphia, Pa. 1886–93, Designing and erecting plant for the manufacture of mineral acids and other chemical products, developing new processes; Chem. Engr., Harrison Bros. & Co., Chem. Works, Philadelphia, Pa. 1893–16, Own firm of Wm. Wilke, Buffalo, N. Y., designing and erecting chemical works; specialty, sulphuric acid, vitric, hydrochloric, acetic, etc. Built plants for Standard Oil Co., Du Pont de Nemours Co., General Chemical Co., Graselli Chemical Co., Penna. Salt. Mfg. Co., Butterworth-Judson Co., Virginia-Carolina Chemical Co., The Phosphate Mining Co., etc.
Present position: Consulting Engineer, American Smelting & Refining Co.

Associate Members

H. H. Allen, Salt Lake City, Utah.
Proposed by J. B. Ambler, George R. Sheldon, C. T. Van Winkle.
Born 1891, Bessemer, Mich. 1905–09, Ishpeming (Mich.) High School. 1909–11, Michigan Agricultural College. 1912–14, Michigan Agricultural College. 1916, Shops of Sullivan Mach. Co., Claremont, N. H. Michigan Agricultural College, B. S. in Engrg. 1915, H. H. Franklin Mfg. Co., Syracuse, N. Y. 1915, Vienna Cons. Mines & Smelt. Co., Hailey, Ida. 1915, Sullivan Machinery Co., Chicago, Ill.
Present position: Sales Engr., Sullivan Mach. Co.

Walter J. Riley, East Chicago, Ind.
Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.
Born 1875, Chicago, Ill. De La Salle Univ. Pres., 1st State Trust & Savings Bank; 1st Calumet Trust & Savings Bank. O. F. Jordan Co., East Chicago, Ind.
Present position: As above.

Paul Stevens, Warren, Ariz.
Proposed by James C. Greenway, Ira B. Joralemon, W. B. Gohring.
Born 1879, Minneapolis, Minn. 1898, Central High School, Minneapolis, Minn. Last eight and one-half years, Private Secy. to Genl. Mgr., Calumet & Arizona Mining Co. Previously in business for self.
Present position: Private Secy. to Genl. Mgr., Calumet & Arizona Min. Co.

Junior Member

Oscar Allen Knight, Cambridge, Mass.
Proposed by Albert Sauveur, H. M. Boylston, F. C. Langenberg.
Born 1893, Athens, O. 1916, Ohio Univ., A. B.
Present position: Student, Mining School, Harvard Univ.

Change of Status—Junior Member to Member

Fletcher H. Wood, Metcalf, Ariz.
Born 1888, Mt. Vernon, N. Y. 1895–1907, Public Schools, Mt. Vernon, N. Y. 1907–09, Sheffield Scientific School. 1912–15, Colorado School of Mines. 1909–12, Costilla Estates Dev. Co. 1913, summer, Mucker, Portland Gold M. & M. Co. 1914, summer, Mucker and Miner, Portland Gold M. & M. Co. 1915, summer, Timber-helper, Portland Gold M. & M. Co. 1915, fall, Engrg. Dept., Arizona Copper Co. 1915–16, Engrg. Dept., Ray Cons. Copper Co.
Present position: Dev. Boss, Arizona Copper Co.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period July 10, 1916 to July 31, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 115, February to August, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of July 31, 1916.

ALLEN, GLENN L................Met., Shattuck-Arizona Copper Co., Bisbee, Ariz.
BABB, P. A....................Ave. Cinco de Mayo No. 32, Mexico City, Mexico.
BAILLIE, FRANK S....................Mgr., Baker Mines Co., Cornucopia, Ore.
BAKER, HAMILTON W.........................1043 Emerson St., Denver, Colo.
BAKER, JOHN M.............................1043 Emerson St., Denver, Colo.
BARKER, L. M...Hayden, Ariz.
BARNES, KENNETT F.....................5151 Washington Ave., St. Louis, Mo.
BARTLETT, WILLIAM SYDNEY, Managing Director, The Copper Mines of Copiapo, Ltd., Copiapo, Chile, South America.
BOWMAN, REGINALD G..Tooele, Utah.
BRIBER, F. E..........................Highland Boy Mine, Bingham, Utah.
BURG, ROBERT S..................American Smelt. & Ref. Co., Durango, Colo.
CAZIN, FRANZ...Herculaneum, Mo.
CHARLES, L. J................................566 High St., Denver, Colo.
CHATIN, AUGUST H.........................3832 Umatilla St., Denver, Colo.
CHEDSEY, WILLIAM R.......................1414 Gaylord St., Denver, Colo.
CRAGIN, RODNEY S..Hill City, S. Dak.
CRANDALL, S. A., Draughtsman, Bunker Hill and Sullivan Smelter, Kellogg, Ida.
DAVIS, LLOYD J...................1010 Boatmen's Bank Bldg., St. Louis, Mo.
DICK-CLELAND, A. F., Dick-Cleland Harper & Co., 49 Jamaica St., Glasgow, Scotland.
DOBSON, CHRISTOPHER G.............Britania Mine, Howe Sound, B. C., Canada.
DOWD, JAMES J.................................Box 389, Houghton, Mich.
FAUST, GUY C........................19 West Jackson St., Wilkes-Barre, Pa.
FIELD, FREDERICK M..............Easton-Pacific Co., Virginia City, Mont.
FOHS, F. JULIUS, Cons. Oil Geol........Suite 307, Gallais Bldg., Tulsa, Okla.
FULLAWAY, RICHARD MERLE...................817 E. 3rd St., Los Angeles, Cal.
GIRAULT, E...........................327 Goliad St., San Antonio, Tex.
GRAYBILL, JOHN H., Industrial Engrg. Dept., Westinghouse Electric and Manufacturing Co., E. Pittsburgh, Pa.
HASTINGS, J. H..........................820 Sheridan St., Monongahela, Pa.
HILL, JOSEPH H...........................Ray Cons. Copper Co., Ray, Ariz.
HLEBNIKOFF, KENNETH I...Goldfield, Nev.
HOSKIN, ARTHUR J..............................P. O. Box 526, Leadville, Colo.
HSU, SHEN-CHIN..........................Songlin, Huchow, Chekiang, China.
HU, S. H.......................................1004 Brush St., Detroit, Mich.
HUDSON, A. W....................Copper Queen Cons. Min. Co., Bisbee, Ariz.
HUGHES, WILSON W..................Monitor Belmont Min. Co., Belmont, Nev.
JOHNSON, C. B................................Federal Lead Co., Federal, Ill.
JOHNSON, J. HARLAN, So. Dakota National Guard, Hospital Corps, Camp Hagman, So. Dak.
JONES, CLEMENS AP-C..Hold publications.
JOPLING, R. F..Hold publications.
KEMP, JAMES T...........................51 Hyatt St., New Brighton, N. Y.
KERNAN, THOMAS H., School of Mines, Experiment Station, Univ. of Minn., Minneapolis, Minn.
KING, PAUL S.........................807 West Ninth St., Wilmington, Del.
KNAPP, GEORGE F....................R. R. 1, Box 67B, West Wrentham, Mass.
LASKY, BERNARD H..............U. S. Smelt. Ref. & Min. Co., Tucson, Ariz.
LAWSHE, VERNER T.....................4819 Warrington Ave., Philadelphia, Pa.
L'ENGLE, E. FLEMING.................................Frisco Bldg., Joplin, Mo.
LESSIG, NICOLAI, Genl. Mgr., Ridder Mining and Trading Co., Ridder Mine, Tomsk Government, Russia.
LEWIS, JOHN MOORE.............Chief Engr., Houston Coal Co., Elkhorn, W. Va.
LEWIS, ROBERT S...........................Box 902, Stanford University, Cal.
LIDDELL, PARKER..Reno, Nevada.

Liebig, J. O. .... 161 Sherman Ave., Newark, N. J.
Lihme, C. B. .... Lake Forest, Ill.
Lindsay, J. A. M. .... Broken Hill Prop. Co., Broken Hill, N. S. W., Australia.
List, E. .... Rolla, Mo.
McDonald, John A. .... Cananea Cons. Copper Co., Cananea, Son., Mexico.
McHardy, R. H. .... General Delivery, Bisbee, Ariz.
MacDonald, Bernard .... Mills Bldg., El Paso, Tex.
MacDonald, Jesse J. .... Box 421, Clarkdale, Ariz.
Mackenzie, George S. .... Craig Lee, Broughty Ferry, Dundee, Scotland.
Malcolmson, J. D. .... Thompson, Nev.
Masters, J. E. .... 434 E. 2d Ave., Tucson, Ariz.
Means, A. H. .... Copper Mountain, Ajo, Ariz.
Millard, William J. .... Box 75, Houston, Tex.
Miller, W. B. .... Sardis, B. C., Canada.
Mosonyi, Emil, Salvador Deep Well Boring Co., P. O. Box 158, San Salvador,
Republic El Salvador, Central America.
Neeld, H. C. .... Hold publications.
Pattinson, R. L. .... Chatham, Ontario, Canada.
Pelton, Harold G. .... Union Oil Co., Avila, San Luis Obispo Co., Cal.
Peschel, William M. .... 51a R. F. D. 2, Lewiston, Ida.
Phillips, William B. .... Austin, Tex.
Pigott, Curtis .... Met., Midvale Smelter, U. S. Smelt. Co., Midvale, Utah.
Rau, H. L. .... Hazel Green, Wis.
Ray, Frank A. .... 515 Huntington Bank Bldg., Columbus, Ohio.
Robinson, E. M. .... Psi U House, S. Bethlehem, Pa.
Roesler, Max, Geol. .... Juragua Iron Co., Box 383, Santiago de Cuba, Cuba.
Rork, Frank C. .... Humboldt, Ariz.
Ruebel, Ernst H., Custodian's House, State Fair Grounds, Springfield, Ill.
Ryder, Thomas J. .... Apartado 189, Vera Cruz, Ver., Mexico.
Schmidt, R. D. .... Corrigan McKinney Co., Cleveland, Ohio.
Schwarz, Charles E., Min. and Met. Engr. .... P. O. Box 715, Miami, Okla.
Skogmark, John .... Room 1509, 35 Nassau St., New York, N. Y.
Smith, Elwyn L. .... Granby, Mo.
Soper, Ellis C., Cons. Engr. .... Mariel, Cuba.
Sproat, A. D. .... 73 W. 5th St., Chillicothe, Ohio.
Steel, Donald .... Douglas, Alaska.
Stroup, Thomas A., Engr. .... Arthur Mine, Utah Copper Co., Garfield, Utah.
Thompson, Warren D. .... Kennebec Hotel, Long Beach, Cal.
Thropp, J. E., Jr. .... Berkshire Iron Works, Sheridan, Lebanon Co., Pa.
Tong, S. K. .... 404 W. 115th St., New York, N. Y.
Tsuboi, Yoshio .... Tsuboi Mining Office, 126, Koyasu, Yokohama, Japan.
Vigeon, E. C., Genl. Mgr., Bede Metal and Chemical Co., Ltd.,
Hebburn, near Newcastle-on-Tyne, England.
Walters, R. E. .... Asst. Mgr., Snow Storm Mines Cons., Troy, Mont.
Watson, Hugh C. .... Yak Mine, Leadville, Colo.
Watt, Arthur P., Met. .... St. Louis Smelt. & Ref. Co., St. Francois, Mo.
Wells, J. S. C. .... Care R. de C. Greene, 211 Broad St., Elizabeth, N. J.
Worth, John G., Min. Engr. .... 836 Real Estate Trust Bldg., Philadelphia, Pa.
Wright, Louis A. .... Casilla 125 D. Santiago, Chile, South America.
Young, Fred W. .... Mercer County Light, Heat & Power Co., Greenville, Pa.

MEMBERS' ADDRESSES WANTED

Name. Last Address of Record from which Mail has been Returned.

Barnes, Blakeslee .... Arrow Engineering Co., Palmyra, Mo.
Bell, D. A. S. .... 136 McLaren St., Ottawa, Can.
Blow, J. J. .... 172 Rodney St., Brooklyn, N. Y.
Booth, E. L. .... 349 W. 145th St., New York, N. Y.
Boys, H. R. .... Aurora Cons. Min. Co., Aurora, Nev.
Cairns, John M. .... Henry S. King & Co., 65 Cornhill, London, E. C., England.
Cole, Robert J. .... McKay Apartments, 7th & Pike Sts., Seattle, Wash.
Crary, Charles N. .... Kimberly, Nev.
Cushing, Daniel .... Sandusky Foundry & Machine Co., Sandusky, Ohio.
Daley, Stephen H., Jr. .... Box 943, Niagara Falls, N. Y.

Draper, Carl H. .... Apartado 77, Guadalajara, Jal., Mexico.
Foster, George C. .... 18 Cadogan Blk., Calgary, Alta., Canada.
Goedicke, Carl .... Box 535, San Antonio, Tex.
Gordon-Firebrace, W. E. .... 812 Salisbury House, London, E. C., England.
Gunther, C. Godfrey .... Stratford, Conn.
Hanlon, John Edward .... Timmins, Ont., Canada.
Hobart, Edmund Norris .... Clifton, Ariz.
Jones, Thomas J., Mine Mgr., Kyshtim Min. Wks., Perm Govt., via Petrograd, Russia.
Kellogg, R. M. .... Wellington, Lyon Co., Nev.
Kishman, Maurice W. .... 101 Masonic Ave., Cripple Creek, Colo.
LeNoir, Frank H. .... Douglas, Alaska.
McGee, John .... United Greenwater Co., Dale via Amboy, Cal.
Mainwaring, H. M. C., Mine Mgr., Chillagoe, Ltd., Chillagoe, Queensland, Aust.
Miller, Frank Barton .... Blair, Esmeralda, Nev.
Mohrman, E. M. .... 1293 W. 111th St., Cleveland, Ohio.
Moore, Redick R. .... Zortman, Mont.
O'Brien, P. C. K. .... Hotel Essex, 684 Larkin St., San Francisco, Cal.
Parrish, S. F. .... Battle Mountain, Nev.
Patchell, F. J. .... 4516 No. Lincoln St., Chicago, Ill.
Paterson, A. W. .... 1814 11th Ave., Spokane, Wash.
Perry, Robert S. .... Kingscourt Apts., 36th & Chestnut Sts., Phila., Pa.
Ralph, E. W. .... Boston Ely Mining Co., Kimberly, Nev.
Redfearn, A. M. .... Hold publications.
Revell, G. E. .... Box 132, Nelson, B. C., Canada.
Reynolds, L. .... Apartado 25, Guanajuato, Mexico.
Rhodes, W. B. .... Golden, Colo.
Rodriquez, J. C. .... Apartado 76, Saltillo, Coah., Mexico.
Rogers, B. C., Care Standard Mine, Detroit Copper Min. Co., Box 427, Metcalf, Ariz.
Sales, A. J. .... Giroux Cons. Mines Co., Kimberly, Nev.
Sandifer, H. C. .... Box 15, Bis., Mexico City, Mexico.
Smith, A. H. .... Gateway, B. C., Canada.
Sparks, J. T. .... 326 Erie St., El Paso, Tex.
Stoddart, A. W. .... 638 Salisbury House, London, E. C., England.
Sullivan, W. P. .... The Henry Walke Co., Norfolk, Va.
Taylor, A. W. .... Korean Exploration Co., Chiksan Mines, Chiksan, Korea.
Thomas, Edmund .... Weaver Mine, Gibson, N. Mexico.
Tod, Grant H. .... Care Coast Mfg. & Supply Co., Livermore, Cal.
Tong, S. K. .... 413 W. 115th St., New York, N. Y.
Van Ness, W. W. .... 622 Salisbury House, London, E. C., England.
Wainewright, W. B. .... 61 Grace Church St., London, E. C., England.
Ward, Harry J. .... 948 Market St., San Francisco, Cal.
Wentworth, I. H. .... 245 Beldon Ave., Harlandale Addition, San Antonio, Tex.
Wraight, E. C. .... 63 Wavertree Road, Streatham Hill, London, S. W., England.
Yeandle, W. H., Jr., Care Guilermo Brockmann, 2a Capuchines,
No. 55, Mexico City, Mexico.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period July 10, 1916 to July 31, 1916.

Date of Election.	Name.	Date of Decease.
1900	*Deidesheimer, P.	—— —, 1916.
1879	**Hinchman, Charles S.	June 3, 1916.
1875	*Kirchhoff, Charles	July 23, 1916.
1914	*Levy, A. G.	—— —, 1916.
1891	**Louis, David A	—— —, 1913.
1907	*Pocock, Cecil W.	July 19, 1916.
1899	*Louis Rosenfeld	July 20, 1916.

*Member. **Life member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.
DAVID H. BROWNE, *Chairman.* PERCY E. BARBOUR, *Vice-Chairman.*
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer.*

Boston

Meets first Monday of each winter month.
W. E. C. EUSTIS, *Chairman.* R. L. AGASSIZ, *Vice-Chairman.*
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.
STANLY A. EASTON, *Chairman.* FREDERIC KEFFER, *Vice-Chairman.*
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
D. C. LIVINGSTON, FRANK A. ROSS.

Puget Sound

Meets second Saturday of each month.
GLENVILLE A. COLLINS, *Chairman.* H. L. MANLEY, *Vice-Chairman.*
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

Southern California

C. COLCOCK JONES, *Chairman.* ALVIN B. CARPENTER, *Vice-Chairman.*
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 216 Union League Bldg., Los Angeles, Cal.
A. B. W. HODGES, R. A. PEREZ,
E. A. MONTGOMERY, WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman.* F. H. BOSTWICK, *Vice-Chairman.*
P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
G. A. KENNEDY. M. S. MACCARTHY.

Montana

J. L. BRUCE, *Chairman.* W. C. SIDERFIN, *Vice-Chairman.*
WALTER E. GABY, *Secretary-Treasurer*, 834 W. Granite St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.
T. A. RICKARD, *Chairman.* W. H. SHOCKLEY, *Vice-Chairman.*
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
E. A. HERSAM, H. W. YOUNG.

Pennsylvania Anthracite

R. V. NORRIS, *Chairman.*
CHARLES F. HUBER, *Vice-Chairman.* EDWIN LUDLOW, *Vice-Chairman.*
W. J. RICHARDS, *Vice-Chairman.* ARTHUR H. STORRS, *Vice-Chairman.*
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP,
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI. *Chairman.* HERMAN GARLICH, *Vice-Chairman.*
F. W. DEWOLF, *Vice-Chairman.* M. M. VALERIUS, *Vice-Chairman.*
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH,

Chicago

CHARLES H. MACDOWELL, *Chairman.* LUTHER V. RICE, *Vice-Chairman.*
HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

Utah

C. W. WHITLEY, *Chairman.* WALTER FITCH, *Vice-Chairman.*
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI, WILLIAM WRAITH.

Arizona

GERALD SHERMAN, *Chairman.*
NORMAN CARMICHAEL, *1st Vice-Chair.* B. BRITTON GOTTSBERGER, *2nd Vice-Chair.*
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
W. L. CLARK, J. C. GREENWAY.
W. G. McBRIDE. FOREST RUTHERFORD.

Nevada

J. W. HUTCHINSON, *Chairman.* FRANCIS CHURCH LINCOLN, *Vice-Chairman.*
HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada.
W. H. BLACKBURN, E. A. JULIAN,
EMMET D. BOYLE, JOHN G. KIRCHEN,
FREDERICK BRADSHAW, C. B. LAKENAN.
TASKER L. ODDIE.

STANDING COMMITTEES

Executive

L. D. RICKETTS, *Chairman.*

GEORGE D. BARRON,
SIDNEY J. JENNINGS,
W. L. SAUNDERS,
BENJAMIN B. THAYER.

Membership

KARL EILERS, *Chairman.*

ARTHUR S. DWIGHT,
LEWIS W. FRANCIS,
LOUIS D. HUNTOON,
ARTHUR L. WALKER.

Finance

GEORGE D. BARRON, *Chairman.*

ALBERT R. LEDOUX,
CHARLES F. RAND.

Library

E. GYBBON SPILSBURY, *Chairman.*[3]

KARL EILERS,[1]
ALEX C. HUMPHREYS,[2]
E. F. ROEBER,[4]
BRADLEY STOUGHTON.

Papers and Publications

BRADLEY STOUGHTON, *Chairman.*

EXECUTIVE COMMITTEE

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SAMUEL A. TAYLOR.

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WILLIAM H. BLAUVELT,
H. A. BRASSERT,
DAVID H. BROWNE,
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ALLAN J. CLARK,
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WALTER E. HOPPER,
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J. E. JOHNSON, JR.,
WILLIAM KELLY,
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CHARLES K. LEITH,
ANTHONY F. LUCAS,
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HERBERT A. MEGRAW,
RICHARD MOLDENKE,
SEELEY W. MUDD,
R. V. NORRIS,
EDWARD W. PARKER,
EDWARD D. PETERS,
R. M. RAYMOND,
ROSSITER W. RAYMOND,
THOMAS T. READ,
ROBERT H. RICHARDS,
L. D. RICKETTS,
HEINRICH RIES,
RENO H. SALES,
ALBERT SAUVEUR,
HENRY L. SMYTH,
A. A. STEVENSON,
RALPH H. SWEETSER,
GEORGE D. VAN ARSDALE,
FELIX A. VOGEL,
ARTHUR L. WALKER,
ROLLA B. WATSON,
HORACE V. WINCHELL,
FREDERICK W. WOOD,
DWIGHT E. WOODBRIDGE.

COMMITTEE ON JUNIOR MEMBERS AND AFFILIATED STUDENT SOCIETIES

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Vice-Chairmen

CHARLES H. FULTON,
EDWIN C. HOLDEN,
FREDERICK W. SPERR,
GEORGE J. YOUNG,

WALTER R. CRANE, *Secretary*, Pennsylvania State College, State College, Pa.

LUTHER W. BAHNEY,
DARSIE C. BARD,
ROBERT H. BRADFORD,
SAMUEL W. BEYER,
GUY H. COX,
JOSEPH DANIELS,
NOAH F. DRAKE,
FRANK W. DURKEE,
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CHARLES E. LOCKE,
JAMES F. McCLELLAND,
HARRY B. MELLER,
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HENRY S. MUNROE,
CHARLES J. NORWOOD,
GEORGE S. RAYMER,
HEINRICH RIES,
HENRY L. SMYTH,
FRANCIS A. THOMSON,
F. W. TRAPHAGEN,
CLINTON M. YOUNG.

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[1] Until Feb., 1917. [2] Until Feb., 1918. [3] Until Feb., 1919. [4] Until Feb., 1920.

COMMITTEE ON INCREASE OF MEMBERSHIP

THOMAS T. READ, *Chairman.*
PHILIP W. HENRY, *First Vice-Chairman.*

Vice-Chairmen

LYNDON K. ARMSTRONG,
F. G. COTTRELL,
CHARLES A. BUCK,
EDWARD L. DUFOURCQ,
H. W. HARDINGE,
H. L. HOLLIS,
DUNCAN MACVICHIE,
HENRY W. NICHOLS,
ERSKINE RAMSAY,
CHARLES F. RAND,
GILBERT RIGG,
WALTER G. SWART,
RICHARD P. TINSLEY.

WILLIAM H. SHEARMAN, *Secretary*, 55 Wall St., New York, N. Y.

HUNTINGTON ADAMS,
F. L. ANTISELL,
EDWIN G. BANKS,
LOUIS V. BENDER,
C. H. BENEDICT,
JAMES G. BERRYHILL,
ALBERT C. BOYLE, JR.,
D. H. BRADLEY, JR.,
VICTOR M. BRASCHI,
FREDERIC K. BRUNTON,
LESTER R. BUDROW,
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M. H. GIDEL,
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RICHARD S. HASELTINE,
FRANK R. HEWITT,
EDWIN HIGGINS,
JOSEPH T. HILLES,
JOHN HOATSON,
ROY J. HOLDEN,
TADASHIRO INOUYE,
YING-CHIEH KUANG,
K. C. LI,
I. P. LIHME,
JOHN J. LINCOLN,
DOUGLAS C. LIVINGSTON,
SPENCER R. LOGAN,
F. E. LUCAS,
WALTER E. McCOURT,
P. M. McHUGH,
FREDERICK J. H. MERRILL,
ARTHUR NOTMAN,
HENRY M. PARKS,
IRVING PERRINE
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Geology of the Warren Mining District

BY Y. S. BONILLAS,* S. B., J. B. TENNEY,* E. M., AND LEON FEUCHÈRE,* INGÉNIEUR CIVIL DES MINES, BISBEE, ARIZ.

(Arizona Meeting, September, 1916)

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I. INTRODUCTION

THE main object of this article is to present the results of observations by the Copper Queen Consolidated Mining Co.'s geological department as an addition to the already published reports on the Warren district. Since these observations relate mostly to the ore deposits, little will be found here on the general geology that has not been covered by F. L. Ransome's admirable work on the Bisbee Quadrangle.[1] A summary of this very important part of the study of the camp will be given, however, mainly to bring out the new facts discovered during the advance in underground development and the latest detailed study of structures and petrography of the mining area.

II. PHYSIOGRAPHY

The Mule Mountains, in which the Bisbee ore deposits occur, constitute a chain with a northwest-southeast axis rising abruptly on the sides from the Sulphur Springs Valley on the northeast, as shown in Plate 1, and the San Pedro Valley on the southwest. The range starts at about the International Boundary near Christianson's ranch, with low hills covering a width of 1 or 2 miles, and extending for about 3 miles to the northwest. Here the Gold Hill overthrust fault has caused an abrupt rise in elevation and from this point on the range widens out and the hills become very much more rugged. The highest point is about 6 miles farther. The total length of the range from the International Boundary is about 23 miles and its maximum width, about opposite the town of Bisbee, is 10 miles. The range finally ends at Government Draw, which is a pass about 2 miles wide, connecting the San Pedro and Sulphur Springs Valleys. Beyond this the Tombstone hills commence.

The range is divided into two parts by Tombstone Canyon, a deep canyon running through the southwest side of the range. It is along this canyon that the Borderland Route road takes its course, and in which the town of Bisbee is situated.

Geologically, also, as seen in Plate 1, the Mule Mountains are roughly divided by Tombstone Canyon into two parts. To the southwest is the pre-Cretaceous tract and to the northeast the Cretaceous. The difference in the two tracts is very marked physiographically. The pre-Cretaceous is very much cut up and is formed of rocks of diverse compositions, consisting of schists, granites, shales, and limestones. These rocks, when subjected to erosion, form a rugged topography with deep canyons and steep cliffs. The Cretaceous tract, on the contrary, is uniform in composition, constituted, for the most part, by soft sandstones, conglomerates, and shales, which are but little faulted. The topography carved from them is characterized by gently sloping hills and draws with almost no cliffs or deep valleys in evidence.

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[1] Geology and Ore Deposits of the Bisbee Quadrangle, *Professional Paper of the U. S. Geological Survey*, No. 21 (1904).

III. INTRODUCTORY GEOLOGY

The oldest rocks of the range are pre-Cambrian schists and an intruded granite, which are separated from the overlying Paleozoic beds by a profound unconformity.

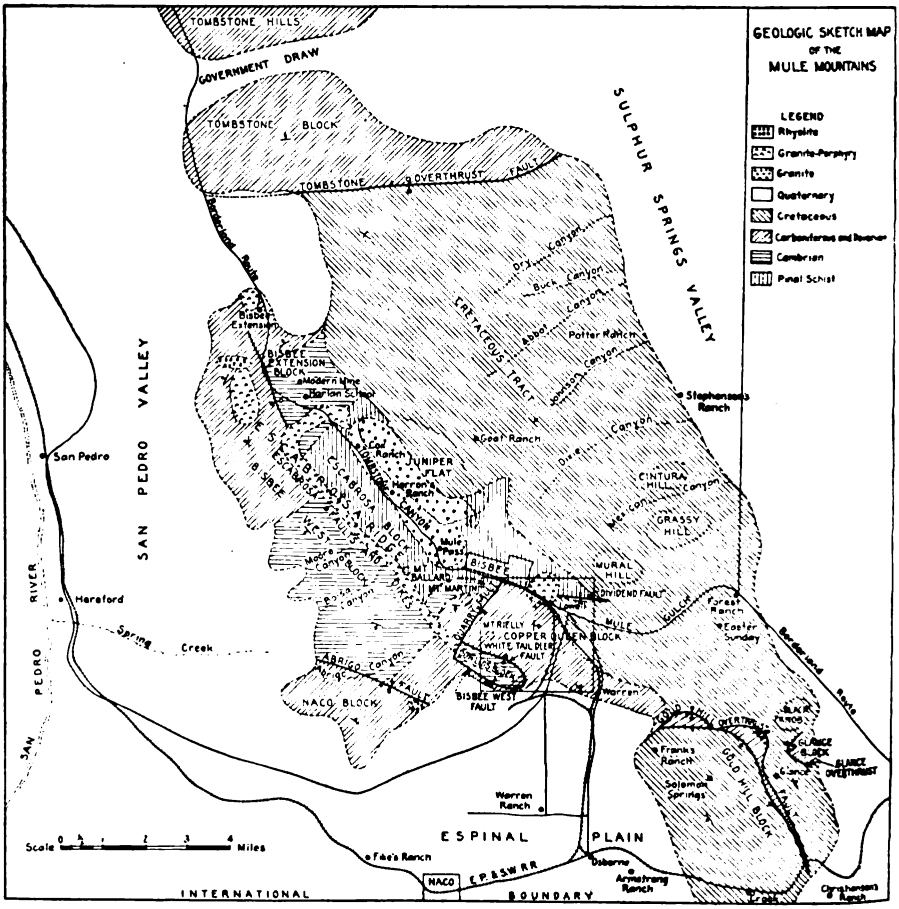

PLATE 1.—GEOLOGICAL MAP OF THE MULE MOUNTAINS.

The Paleozoic beds represent an era of apparently uninterrupted deposition of sediments, starting with 440 ft. of quartzites having a basal conglomerate, followed by Cambrian limestones, Ordovician-Silurian quartzite, Devonian and Carboniferous limestones. The total thickness of the Paleozoic beds is approximately 5,000 ft.

At the end of the Carboniferous era a violent uplift took place accompanied by extensive faulting and intrusion of granite porphyry. During this uplift, and intimately associated with the intrusion, mineralizing solutions arose from which originated the orebodies of the camp.

Following this came a long period of erosion, which ended with a rapid subsidence, during which time 4,500 to 5,000 ft. of Cretaceous sandstones, shales, and limestones were deposited.

At the end of Cretaceous time a gentle uplift took place, lifting the range again above sea level. This was accompanied by some intrusion of rhyolite and monzonite.

During Tertiary and Quaternary ages, this land area was subjected to erosion, resulting in the present topography, the detritus filling in both the valley of the San Pedro and of the White River, one on each side of the range.

IV. ROCKS OF THE DISTRICT

A. SEDIMENTARY ROCKS

Reference to Plate 2, showing a generalized geologic section, will help make clear the following description.

Pinal Schist

Name.—The oldest rock of the district is the Pinal Schist. It is composed of a uniform series of thinly laminated siliceous mica schists of unknown thickness. From the similarity in texture and stratigraphic relation to the later sediments, it has been correlated by Ransome[2] with the underlying schistose complex of the Pinal Range, which he called the Pinal Schist.

Distribution and General Structure.—The Pinal Schist, being the basal crystalline rock upon which rest all the younger formations of the district, is exposed everywhere that erosion has stripped them off, and is encountered underground on sinking through them.

The best surface exposures are northeast of Mule Gulch and to the southwest of Tombstone Canyon. Smaller exposures are also found about 1 mile to the northwest of the town of Don Luis. Underground it is exposed where the Dividend Fault is cut, both in Copper Queen and in Denn ground. It also is found as dragged-in fragments in the contact-breccia mass around Sacramento Hill.

Lithology.—The color varies from light to dark gray, with tinges of green on fresh surfaces, and rusty in weathered specimens. The cleavage is rather imperfect, having a shiny satin-like surface.

The microscope shows the rock to be composed mainly of quartz and sericite. Sometimes the sericite is replaced in part by penninite, chlorite, or serpentine. Folded into the schist, and forming an integral part of it, are occasional masses of diabase.

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[2] *Loc. cit.*, p. 24.

Origin and Age.—The age of the Pinal Schist is assumed to be the same as for the Pinal Schist of the Globe district which Ransome[3] has shown to be derived from arenaceous sediments, probably of Algonkian age. Both at Globe and at Bisbee a profound unconformity exists between it and the overlying Paleozoic beds.

COLUMNAR SECTION

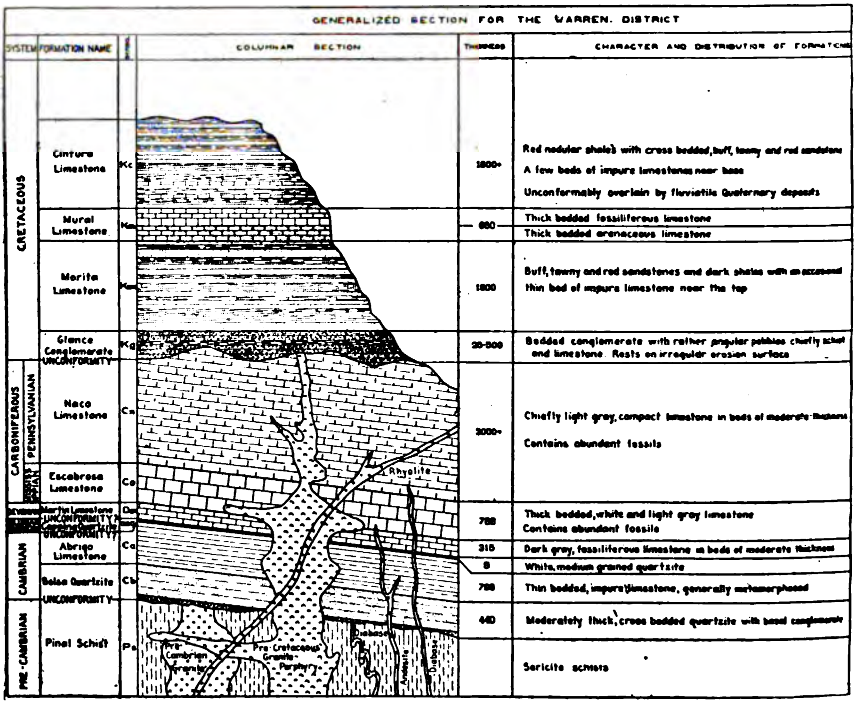
GENERALIZED SECTION FOR THE WARREN. DISTRICT
SYSTEM
FORMATION NAME
COLUMNAR SECTION
THICKNESS
CHARACTER AND DISTRIBUTION OF FORMATIONS
CRETACEOUS
Cintura Limestone
Kc
1800+
Red nodular shales with cross bedded, buff, tawny and red sandstone
A few beds of impure limestones near base
Unconformably overlain by fluviatile Quaternary deposits
Mural Limestone
Km
650
Thick bedded fossiliferous limestone
Thick bedded arenaceous limestone
Morita Limestone
Km
1800
Buff, tawny and red sandstones and dark shales with an occasional thin bed of impure limestone near the top
Glance Conglomerate
UNCONFORMITY
Kg
20-500
Bedded conglomerate with rather angular pebbles chiefly schist and limestone. Rests on irregular erosion surface
CARBONIFEROUS
PENNSYLVANIAN
Naco Limestone
Cn
3000+
Chiefly light grey, compact limestone in beds of moderate thickness
Contains abundant fossils
Escabrosa Limestone
Ce
Martin Limestone
UNCONFORMITY?
Capping Quartzite
UNCONFORMITY?
Dm
700
Thick bedded, white and light gray limestone
Contains abundant fossils
CAMBRIAN
Abrigo Limestone
Ca
315
Dark grey, fossiliferous limestone in beds of moderate thickness
8
White, medium grained quartzite
Bolsa Quartzite
Cb
790
Thin bedded, impure limestone, generally metamorphosed
UNCONFORMITY
PRE-CAMBRIAN
Pinal Schist
Ps
440
Moderately thick, cross bedded quartzite with basal conglomerate
Sericite schists
Rhyolite
Pre-Cambrian Granite
Pre-Cretaceous Granite-Porphyry
Diabase
Andesite
Diabase

PLATE 2.—GENERALIZED GEOLOGIC SECTION SHOWING ROCK STRATA OF THE WARREN DISTRICT.

Bolsa Quartzite

Name.—The name of this formation, as well as those for all other formations in the district, were given by Ransome[4] and have been generally adopted.

Distribution and General Stratigraphy.—This Bolsa Quartzite lies on an evenly eroded plain of Pinal Schist. The main exposures of this formation are southwest of Escabrosa Ridge, especially in Bolsa, Abrigo, and Quarry Canyons. In the last-named canyon a complete unfaulted section occurs. Underground, the formation has been cut by diamond-drill holes from the bottom of the Irish Mag shaft, and also by the workings from the Wade Hampton shaft.

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[3] *U. S. Geological Survey*, Globe Folio, No. 111, p. 2.

[4] *Loc. cit.*, p. 28.

Lithology.—At the base of the formation is a well-marked basal conglomerate, much iron-stained, containing pebbles consisting mostly of white, well-rounded quartz of up to a 4-in. diameter, and averaging about 1 in., the cementing material consisting of subangular sand. In the northwestern part of Tombstone Canyon some pebbles of porphyritic rock are found, together with some rounded crystals of feldspar. These feldspar crystals consist of pink microcline and microperthite, such as are found in the Juniper Flat granite less than a mile away. These feldspars, notwithstanding their being subject to such long exposure, are still relatively fresh.

The next 100 ft. of the Bolsa shows alternating beds of fine conglomerates and quartzites, with some arkose beds. Then follows a finer whitish quartzite with frequent cross-bedding, succeeded by some massive, dense beds of a maroon-colored quartzite. The total thickness of the Bolsa Quartzite as measured in Quarry Canyon is 440 ft.

Age.—No fossils have been found in the Bolsa except sparse worm tracks near the top, but as the next formation lies conformably on top of it and shows middle Cambrian fauna, the Bolsa has been assumed as lower or middle Cambrian.

Abrigo Limestone

Distribution and General Stratigraphy.—The Abrigo formation follows the Bolsa Quartzite conformably, and is the first fossil-bearing formation in the district.

The best exposure of Abrigo limestone is found on the northern slope of Mount Martin, where a complete unfaulted section is obtainable. Large areas of Abrigo are also found on the southwestern slope of Escabrosa ridge in Abrigo, Moon, and Bolsa Canyons, but the formation is here badly faulted. Small areas are also found in Quarry Canyon, north of Don Luis, and on the southern slope of the range from Mount Reilly to Gold Hill. As the beds are, for the most part, soft shales and shaly limestones, good exposures are rare, the formation being usually covered by talus.

Underground, the Abrigo formation is cut by the lower levels of all the mines from the Higgins to the Briggs, but in most of them only the top 100 ft. of rock has been cut. At the Spray, Oliver, and Irish Mag shafts, however, the formation has been cut by their lower workings to a depth of 400 ft. below its top, and diamond-drill holes from the Irish Mag shaft have penetrated into the Bolsa. At the White Tail Deer and Wade Hampton shafts the lower beds have also been cut. No complete section, however, has been exposed underground. Measured at Quarry Canyon the thickness of the Abrigo was found to be 798 ft.

Geological section of Abrigo at Quarry Canyon:

Thin shales, greenish-gray to pink-brown, highly epidotised	0–72
Gray broken calcareous limestone, epidotised	72–83
Shaly calcareous sandstones	83–112
Medium-grained greenish-gray limestone with epidotised beds	112–132
Pure fine-grained pinkish-gray limestone	133–134
Arenaceous pinkish-gray shales	134–136
Purer fine-grained dark-gray limestone	136–139
Light-green shales	139–141
Fine-grained limestone, thin wavy bands of epidote	141–146
Green shales	146–151
Succession of impure lime and shales	151–183
Fine-grained gray limestone, beds of epidote ½ to 1 in. thick	183–229
Purer light-gray limestone	229–245
Coarse sandy limestone with epidotised beds	232–245
Thin greenish shales	245–253
Coarse limestone, very fossiliferous (worm tracks)	253–261
Arenaceous green limestone	261–278
Coarse limestone, thin epidotised bands, some purer, 2-ft. beds forming steps on surface	278–412
Impure limestone green to brown, thin wavy bands of epidote very characteristic on surface by the weathering out of the epidote	413–497
Shaly arenaceous limestones	497–521
Coarse gray arenaceous limestone	521–629
Coarse light-gray crystaline sandy limestone	629–684
Coarse cherty-banded crystalline white bed-forming cliff	684–700
Sandy even-grained limestone and quartzite	700–798

The Abrigo formation consists of a series of shales and argillaceous and sandy limestones, with bands of epidote. These epidote bands weather out on exposure to the atmosphere, giving the formation a characteristic cherty banded appearance. In the top beds there are a few bands of true chert. Underground, this epidote appears as greenish bands giving a wavy banding which readily distinguishes it from all other formations, when the beds are fresh. About 100 ft. from the top there exists a 16-ft. bed of pure white limestone which forms a rather prominent cliff when encountered on the surface.

Capping Quartzite

Lying on top of the Abrigo, and with no apparent angular unconformity, is a persistent bed of pure quartzite varying in color from white to dark red, and varying from 6 in. to 12 ft. in thickness, and locally called the Capping Quartzite.

Distribution.—This quartzite being more resistant than the underlying and overlying beds, is exposed wherever the Abrigo outcrops, as a small cliff or shelf. One well-marked exposure is found to the east of Black Gap, where it runs up the side of the hill apparently as a siliceous dike. Other good exposures are found on Mt. Martin, and southwest of Escabrosa Ridge.

Underground, this bed is important as a marker. Due to its character, it is resistant to metamorphism, and may be recognized easily even when the rest of the sediments have been completely changed. Its thinness, however, tends to make it elusive, as a small fault frequently will throw it out of an otherwise perfect section.

Age.—As one travels north through the State, this formation becomes thicker, and in the Globe district it attains a thickness of 500 to 700 ft., while that of the underlying limestone has decreased to 200 ft. Here there is a well-marked evidence of an unconformity between the limestone, which is capped by a flow of basalt of varying thickness, and the overlying Troy Quartzite, which is conglomeratic at its base. While no fossil evidence is available, its relation to the overlying and underlying formations seems to indicate that it is probably of Silurian age, and that the Ordovician is represented by a period of erosion, when the area studied was very little above sea level, since there is no apparent angular unconformity below or above this formation.

Martin Limestone

Lying conformably above the Capping Quartzite is 300 ft. of limestone known as the Martin limestone.

Distribution.—The exposures of the Martin limestone are widespread, especially along Escabrosa Ridge, north of Moore Canyon, in Abrigo Canyon, Escacado Canyon, and on the southern slopes of the hills east of Mt. Reilly. Owing to the fact that the overlying Escabrosa limestone is a prominent cliff-forming bed, the exposures are apt to be talus-covered. The best complete sections are obtained on the northern slope of Mt. Martin, and to the east of Black Gap. Underground, it is the formation most frequently cut of any in the ore zone. For this reason it is usually difficult to recognize from its lithological characteristics. The formation is so rich in fossils, however, that even when much altered, a diligent search will yield some trace of them. The formation is encountered underground in every mine in the district except the Wade Hampton, which is entirely in Cambrian, and the Denn which did not sink deep enough to strike it. The outcrop of this formation was first found in the Queen opencut near its top, and it has been encountered at progressively lower levels from here down to the 1,600-ft. level of the Lowell, and the 1,500-ft. level of the Briggs.

Lithology.—The Martin limestone is the most constant formation, both in respect to thickness, and character, of any in the district. Plate 3 shows a comparison of sections obtained over widely scattered parts of the district. As is seen, the formation as a whole is made up of fairly dense dark-gray limestone with occasional shaly and sandy members, but all with enough lime present to be called limestone. It is all more or less fossiliferous, and some beds are extremely so. The characteristic fossils are three brachiopods, one a spirifer, one an atrypa, and one a schisophoria, which all attain a length of about ⅓ to ½ in.; they tend to weather out on both surface and underground exposures. When the rock is altered, these fossils tend to resist metamorphism to a remarkable degree, and it is not at all rare to find almost perfect fossils left in a soft mass of completely altered limestone. These fossils, however, are frequently found completely pyritized when the surrounding

CORRELATION OF DEVONIAN BEDS

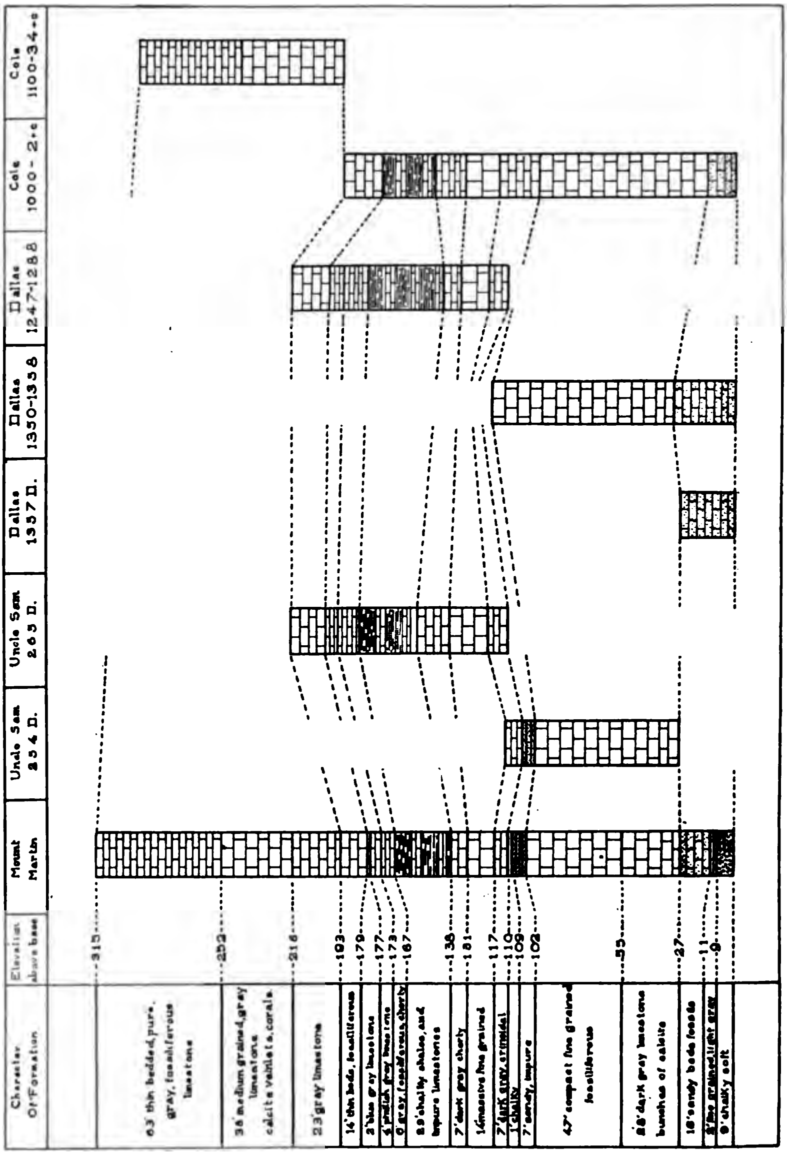

Plate 3.—Comparison of Sections of Martin Limestone in Various Parts of the Warren District.

limestone is untouched. In one case, within a completely oxidized orebody of the Martin, fossils were found almost perfect in form but completely replaced by azurite. Fortunately, the fossiliferous beds themselves seem to be more easily replaced by ore solutions, so that where metamorphism is most complete, some fossil evidence may usually be found.

The upper beds of the formation have few brachiopods, but are almost made up of corals. On altering, these beds tend to take a pseudo wavy banding, which makes it difficult to distinguish from the Abrigo. The succession of beds in these cases is the only determining factor.

The Martin limestone is not a pure limestone by any means. Almost the whole series is more or less dolomitic, more so than any other formation in the range. It is also much more aluminous than a superficial examination would lead one to suppose. The silica content is also high. Characteristic partial analyses of unaltered specimens of Abrigo and Martin limestones from both surface and underground exposures are given in Table 1. These determinations were made at the Copper Queen assay office.

TABLE 1.—*Analyses of Unaltered Abrigo and Martin Limestones*

	Abrigo					Martin							
	Mt. Martin Section Underground												
	Shales	72–232 ft. above base	232–521 ft. above base	521–700 ft. above base	Cherty Limestone 1,357 Drift	Black Gap[4]	500 Level Uncle Sam Mine[3]			265[3] Drift	1,300[1] Dallas	1,200[2] Dallas	1,000[5] Cole
SiO_2...	68.6	30.8	35.3	16.1	49.4	10.3	39.4	46.0	21.7	15.5	45.5	35.5	25.2
Al_2O_3...	9.1	4.0	3.0	2.1	4.5	2.0	9.1	10.8	6.5	3.5	12.9	5.0	5.9
MgO...	4.6	3.6	2.3	1.7	2.4	14.2	8.8	4.4	13.8	22.9	3.8	5.4	8.5
CaO...	9.8	33.5	31.5	38.9	17.8	28.5	6.0	10.5	17.8	10.7	13.1	26.7	24.7
Fe.....	5.1	2.7	2.8	1.5	2.2	1.2	3.7	3.2	2.0	1.8	3.0	2.1	1.8
S......	...	....	....	0.2	0.6	0.2	1.5	0.4	0.7	1.3	1.9	0.8	0.8
Mn....	...	....	....	....	....	....	0.6	0.4	0.4	0.3			

[1] Bottom beds. [2] 100 ft. above Capping Quartzite. [3] Somewhat metamorphosed. [4] Composite of section. [5] 200 to 300 ft. above Capping Quartzite.

As is seen, the two formations are of about the same purity, but the Abrigo contains more silica than the Martin. The Martin, however, is much more dolomitic, and the lower beds are very shaly. The composite from Black Gap shows that on the whole it is a fairly pure dolomite, with only 13 per cent. of silica, alumina, and iron. The impure beds are chiefly at the bottom 100 ft. and the top 50 ft. of the formation.

Age.—The age of the Martin has been well established by good fossil evidence as Devonian. The best exposition on the correlation of these beds is found in the previously mentioned Ransome's report on the Bisbee Quadrangle.

Escabrosa Limestone

Distribution.—The best surface exposures of this formation are on Escabrosa Ridge from Mt. Martin to Mt. Reilly, and on the southern slopes of the hills from Mt. Reilly to Gold Hill. The formation again appears to the northward on Escabrosa Ridge southwest of the Modern mine, and an excellent section is obtained from where the Borderland Route turns out of Tombstone Canyon into the San Pedro Valley. Underground, the formation is cut by all the shafts from the Czar to the Briggs. It is usually in the oxidized portions of the mines, and as a consequence exposures are not very good.

The best exposures found are at the Junction and Sacramento mines. At the Junction, workings cut the formation from the top to the bottom. Here, except for marbleization, the exposures are good, as oxidation has not penetrated nearly as deep as in the other mines of the camp. At the Sacramento, exposures are obtained of the bottom 300 ft., but marbleization has so altered the formation that they are not good.

Lithology.—The Escabrosa and Naco formations represent really one period of deposition during Carboniferous time. The division of the formation as a whole into Escabrosa and Naco is largely an arbitrary matter. Roughly it is divided, from the fossil evidence, into Mississippian and Pennsylvanian, the former locally called Escabrosa, and the latter Naco. This line of subdivision lies at approximately 800 ft. above the top of the Martin formation. In this paper the thickness of the Escabrosa has arbitrarily been taken as 768 ft. irrespective of fossil evidence. Following is a detailed section of Escabrosa taken at Mt. Martin and northeast of the White Tail Deer shaft.

	Feet
Coarse crystalline pink non-fossiliferous limestone	0–6
Pinkish-white to gray medium-grained 6-in. to 12-in. beds	6–60
Massive grayish-white coarse fossiliferous cliff beds	60–140
Thin gray beds medium fine-grained fossiliferous	140–145
Coarse massive grayish fossiliferous cliff bed	145–185
Thin fine-grained fossiliferous gray beds	185–312
Fine-grained grayish-white cherty banded (½ in. to 3 in.)	312–479
Coarse massive white cliff beds fossiliferous	479–556
Fine-grained thin-bedded fossiliferous	556–608
Brown cherty fossiliferous limes	608–616
Grayish fine-grained thin-bedded fossiliferous	616–648
Pinkish beds medium-grained with some chert	648–686
Fine-grained thin-bedded soft grayish	686–758
Massive dark-gray medium-grained	758–768

The Escabrosa limestone rests conformably on the Martin limestone wherever exposed. The 6-ft. pink non-fossiliferous bed forming the bottom member serves in unaltered ground as an excellent marker. The pure coarse crystalline bed following is the most commonly exposed member of all, forming as it does the steep and rugged cliffs of Escabrosa and Higgins Ridges. At the Mt. Martin and the White Tail Deer sections, it contains very little chert and abundant crinoid stems. Underground exposures appear to contain considerably more chert in this horizon. In fact, the presence or absence of chert beds seems to be very local, and untrustworthy in identifying any particular horizon. Above this first 180-ft. cliff-forming horizon is a soft series of more dense fine-grained beds with some chert, followed by 127 ft. of extremely cherty banded medium-grained limestone, which seems to be very persistent. Above this is a second thinner cliff-forming horizon 77 ft. thick, followed by dense, relatively impure limestone, varying in color from brown to gray with quite abundant chert nodules. There is no definite marker between the Escabrosa and Naco. As previously stated, the Escabrosa is given arbitrarily a thickness of 768 ft.

Naco Limestone

Distribution.—This limestone is generally exposed on the top of the hills from Queen Hill to the east, and extends over the productive area from the Irish Mag shaft continuously east until covered by the later conglomerate. Good exposures of Naco are also found at the northermost range of hills just south of Government Draw, and on Escabrosa Ridge from the Modern mine, to the northwest, where it is finally covered by Quaternary wash. The best and least faulted section of the formation is found in the Naco Hills about 3 miles west of the town of Don Luis. Underground it is cut by all the shafts from the Irish Mag eastwards. Practically no drifts, however, have been driven in this horizon, except possibly at the upper levels of the Junction mine, and the 200-ft. level of the Lowell.

Lithology.—A thickness of 1,500 ft. has been measured in the Naco Hills. Assuming that the formation has been half stripped off we get a probable former thickness of at least 3,000 ft. Compared with the Escabrosa limestone, it is generally thinner bedded, and very much finer grained, and contains some distinctly shaly members, especially in the upper portions. Some coarse-grained beds do exist, however, which without fossil evidence would be impossible to distinguish from the Escabrosa. Abundant crinoids exist as in the Escabrosa, but large brachiopods and cephalopods are much more in evidence. The most common and characteristic Naco fossil is a large-sized member of the productus family.

Cretaceous Deposits

Glance Conglomerate.—Resting on an extremely uneven erosion surface of Paleozoic beds, schist and granite, is deposited the Glance Conglomerate. This formation is exposed over almost the whole of the southern end of the range.

Lithology.—This conglomerate is made up of partially rounded fragments of schist, porphyry, both fresh and sericitized, quartzite, and limestone. These fragments vary in size from ½ in. to 3 ft. diameter and are loosely compacted. Almost no cementing material exists. The conglomerate as a whole is stained a deep brown. The thickness varies from nothing up to 1,000 ft., depending on the old erosion surface. Where this old surface was fairly well worn down, as north of the Dividend Fault and on top of the Juniper Flat granite mass, the formation is a typical basal conglomerate with rounded pebbles cemented with sand.

Age.—The age of the formation is either late Jurassic or early Cretaceous.

Morita, Mural and Cintura Formations.—For a more detailed description of the Cretaceous beds above the Glance Conglomerate, Ransome's report may be referred to. As these beds occur outside the productive area, no additional study was made of them. The Morita consists of a series of buff and tawny sandstones, shales and calcareous sandstones, containing sparse fossil remains, and measuring 1,800 ft. in thickness. This is overlain conformably by 650 ft. of limestone with a 50 to 200-ft. member of pure limestone containing abundant Cretaceous fossils.

The Mural is followed by the Cintura formation, consisting of beds almost identical with those of the Morita, with a thickness of 1,800 ft., plus an unknown amount eroded off. These Cretaceous beds cover about two-thirds of the range, nearly the whole area northeast of Tombstone Canyon and Mule Gulch, and are finally buried beneath the Quaternary wash of Sulphur Springs Valley.

B. IGNEOUS ROCKS OF THE DISTRICT

The most important igneous rocks of the district, those of granitic composition, have been grouped together in the United States Geological Survey publications on this region, and have all been ascribed to the same magma and general period of eruption, although differences in composition, texture and field relations were noted. Notman[5] has already separated the granite and the porphyritic rocks by placing the first in the pre-Cambrian, and Boutwell has found a basic variety of porphyry underground and on surface, which corresponds probably to the post-Cretaceous dike described by Ransome from the Glance mine,[6] this latter being a rock easily distinguished from the other porphyry.

Careful search for field and petrographic evidence has yielded results which allow the following classification of the igneous rocks of the Warren district.

Pre-Cambrian Granite

On Juniper Flats and along the north side of Tombstone Canyon, a mass of granite is exposed which forms the center of the Mule Mountains. The rock is a pink or purplish-gray coarse-grained granite whose component minerals are quartz, orthoclase, microcline, a little biotite and plagioclase feldspar. Accessory minerals can be seen only under a microscope in thin section, such as apatite, magnetite, zircon and tourmaline. The orthoclase contains microperthitic intergrowths of albite, and dusty inclusions of iron oxides.

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[5] *Transactions of the Institute of Mining and Metallurgy*, vol. 22, pp. 550–562 (1913).

[6] *U. S. Geological Survey, Professional Paper* No. 21, p. 84 (1904).

This rock corresponds exactly to that described by Ransome as the Juniper Flats Granite, an analysis of which is given in *Professional Paper* No. 21 *of the United States Geological Survey,* as follows:

	Per Cent.		Per Cent.
SiO_2	75.86	Na_2O	3.60
Al_2O_3	12.17	K_2O	5.04
Fe_2O_3	0.85	H_2O-	0.27
FeO	0.36	H_2O+	0.72
MgO	None	T_1O_2	0.21
CaO	0.62		
			99.70

There are on Juniper Flats some porphyritic alaskite dikes that seem to be directly related to the granite, but aside from these, all other porphyritic rocks here have not been ascribed pre-Cambrian age.

Field Relations.—The Juniper Flats granite has been found cutting only Pinal Schist, and while it has not been found directly overlain by Paleozoic sediments but covered directly by Cretaceous, pebbles of microcline and of microperthite granite have been found at the base of the Cambrian quartzite. This granite nowhere cuts the Paleozoic sediments, but it is itself extensively intruded by dikes and irregular masses of the later porphyries that do penetrate the sedimentaries. These observations, as well as the similarity of this granite to other pre-Cambrian granites in Arizona has led to the belief that the intrusion took place before the beginning of the Paleozoic.

In the area economically exploited in the district for copper, the Juniper Flats granite has not been found, and the only ore associated with the rock comes in fissure veins filled with quartz and some fluorite with pockets rich in gold.

Pre-Cretaceous Granite Porphyry

The most important igneous rock in the district is the granite porphyry that is associated with most of the orebodies, and whose largest outcrop is the mass of Sacramento and Copper King Hills in Bisbee.

Distribution.—On surface this rock is found abundantly in the southwest, or older portion, of the Mule Mountains, but the distribution is by no means even. In the schist, granite, and Cambrian formations there are abundant masses of porphyry in the form of dikes and sills, while in the Carboniferous only a few well-marked dikes are found besides the main stock of Sacramento Hill.

In Juniper Flats and on Escabrosa Ridge there is abundant granite porphyry, some of the dikes cutting granite, schist and Paleozoics in their course from the north to the south side of Tombstone Canyon. On one of the ridges close to the mouth of Tombstone Canyon, about opposite the old Modern mine there is a continuous sill of granite porphry several hundred feet thick in Carboniferous limestones. This sill is remarkable because it is cut by dikes of rhyolite, and because on an old erosion surface of the granite porphyry and Carboniferous limestones a considerable thickness of rhyolitic lavas has accumulated.

The character of these later dikes and flows is exactly the same as that of the volcanic plug with flow structure which is found in the Naco hills and is mentioned by Ransome in his report on the district. The relations in the locality at the mouth of Tombstone Canyon leave little doubt as to the great difference in age between the granite porphyry and the rhyolite, as the erosion surface on which the rhyolite flowed is probably post-Cretaceous.

In the mining area of the district the Sacramento Hill stock has penetrated between the Paleozoic sediments on the south side of the Dividend fault and the schist on the north. The outline of the stock is irregular, and extends about a mile from north to south and an equal distance along the fault. The only other porphyry outcrop of any size in the mining area is the Shattuck dike, which runs from above the Spray mine for more than a mile westward. Another important dike, the Sacramento, outcrops a few isolated places in the Naco limestone, southward from Sacramento Hill almost to the Briggs mine.

In the underground mine workings, the relative abundance of porphyry intrusions in the formations lower than the Carboniferous holds about the same as on surface. Besides the main mass of the Sacramento Hill intrusion there is another smaller, laccolithic mass of porphyry around the Lowell mine. This body is connected with Sacramento Hill and Sacramento dike by masses of porphyry, none of which reach the present surface, but which spread out mostly at the Carboniferous-Devonian horizon. This mass, with its accompanying contact breccia is shown in Plate 6.

The mode of intrusion of the granite porphyry and its relation to the structure of the older rocks will be touched upon later. It will be enough to state here, however, that the intrusions took place in one general period, but not all at once, nor were the conditions the same at the beginning and end of the eruptions, or the composition of the rock exactly the same. Considerable alteration was also started in the rocks before the last porphyry was injected.

Lithology.—Although there is abundant fresh granite porphyry in the district most of the rocks around the ore zones have been so altered by mineralizing solutions that it is difficult to find material which represents the original granite porphyry of the main masses associated directly with the ores. Fortunately, the alteration processes have been so varied that they have sometimes left untouched some of the phenocrysts, at other times the base, so that from various sources a reconstruction can be made of what was for a short time the rock of Sacramento Hill and the Lowell mass. Of the dikes fresh rock is available.

The pre-Cretaceous granite porphyry is, when fresh, a light-gray rock weathering to a light-red color on surface. It has abundant phenocrysts, up to ½ in. in diameter of feldspar, quartz and biotite, the relative amounts of these being variable. The base is a fine-grained mass, almost glassy at times and composing from one-half to about eight-tenths of the rock. The microscope shows that the rock varies from a typical granite porphyry to a rather basic monzonite porphyry, sometimes in the same sill or dike. The main masses were probably of the more acid variety, with well-developed quartz phenocrysts that have embayments due to reabsorption. There are no noticeable inclusions in the quartz. In the more basic varieties of the porphyry, quartz phenocrysts become extremely scarce.

The feldspar phenocrysts are mostly orthoclase with albite and oligoclase increasing as the rock becomes more basic. The orthoclase is in large crystals when it predominates and in small ones when plagioclases are abundant. The only intergrowth of the orthoclase seen is in the form of a micropegmatite with quartz, and at the rim of some large crystals in the dikes where an aureole of quartz and orthoclase usually forms around a pure orthoclase of quartz center. The plagioclases are all between albite and oligoclase and sometimes are in greater abundance and size than the orthoclase.

Biotite is the only ferromagnesian mineral of any importance and comes in large well-formed crystals as well as fine shreds. In most of the porphyry it has been altered before any other minerals, and its traces cannot be distinguished megascopically, but in some of the more basic varieties, which are also chloritized, biotite is the most conspicuous phenocryst mineral and gives a decidedly dark color to the rock.

The base of the porphyry is usually a cryptocrystalline mass of quartz and feldspar, found sometimes with a micropegmatitic intergrowth, at other times with incipient spherulitic growth and a few times with a holocrystalline but extremely fine crystallization.

As accessories, there are: apatite in well-formed needles sometimes noticeably abundant, and generally in the biotites; very few grains of pyrite, and dusty undeterminable inclusions in the feldspars.

The foregoing description of the unaltered porphyry is rarely applicable to the rock encountered in the mining area, as the porphyry may be found in the form of a white mass of sericite and quartz, or a dense dark-green rock composed mostly of serpentine and penninite, with all variations between. These characteristics will be considered later under metamorphism and mineralization.

Age.—The granite porphyry distinctly cuts the Pinal Schist, granite and all the Paleozoic sediments, while pebbles of the altered sericitized and oxidized porphyry are found at the base of the Cretaceous Glance Conglomerate. It is, therefore, certain that the age of intrusion is post-Carboniferous and pre-Cretaceous. That the porphyry intrusion did not take place all at once, is shown by dikes of the slightly more basic varieties cutting the contact breccias of the more acid ones. This difference in age is further emphasized by differences in alteration, but, as will be shown later, there is no reason to believe that there was any great interval of time between the porphyry injections, but that they all belong to the same general period.

Rhyolite

This rock has not been distinctly recognized in the mining area of the Warren district, but due to its great similarity to the granite porphyry it may have been confused with the older rock, especially where both are found in dikes. The rhyolite from the mouth of Tombstone Canyon, whose field relations have already been given, is a gray to pink rock with decided flow structure in its surface forms. Phenocrysts are not abundant and are seen under the microscope to be entirely quartz and orthoclase, in an aphanitic or cryptocrystalline groundmass. Some of the dikes on Escabrosa ridge, and others northeastward from the mouth of Tombstone Canyon, have such a glassy groundmass, and so few phenocrysts of quartz and orthoclase, that they may very well be of the late rhyolite, especially as they are seen to cut other porphyritic dikes in places. The rhyolite of the Naco Hills described by Ransome is probably of the same age as that at the mouth of Tombstone Canyon.

Age.—Unfortunately no information is available at present for placing the age of the rhyolite any closer than post-granite porphyry. An erosion period separates the two, but whether it is the pre-Cretaceous or post-Cretaceous one cannot be definitely stated.

Other Dike Rocks

Hornblende Andesite.—At a few scattered places on the surface, and in the Shattuck, Cole, Wade Hampton and Wolverine mines, there are dikes of a dark-green, fine-grained rock, which is seen under the microscope to be composed of abundant small phenocrysts of andesine and labradorite and a few of hornblende, in a microcrystalline base of feldspar and fine magnetite grains. This andesite cuts all the pre-Cretaceous formations, including the granite porphyry, but its relation to the ores is not clear. The rock is considerably altered, and chloritic minerals especially penninite, as well as some pyrite, have been developed. In a specimen from the Wolverine shaft there is also some sericitization, so it is possible that the andesite may antedate the last of the mineralization period.

Diabase.—Olivine and augite diabase in small masses and dikes have been found to belong to two distinct periods. The earlier is folded in with the Pinal Schist and is pre-Cambrian, while the latest cuts the Carboniferous limestones and is entirely unaltered. There is no known exposure of them underground.

Monzonite Porphyry.—The dikes of this rock mentioned by Ransome as occurring around the Glance mine have not been studied during the preparation of this report, and no similar rock has been found in the mining area, unless it is the one classified as andesite porphyry, or the basic porphyry found by Boutwell.

V. STRUCTURAL GEOLOGY

General Structure of Mule Mountains

Fault Blocks and Other Divisions

In this report the structural geology will be confined almost entirely to the southwestern part of the range covered by pre-Cretaceous sediments, as these play the most important part in the relation to the orebodies of the district. Reference to the areal geology map, Plate 4, and sections made along its lines *A–A*, *B–B*, and *C–C* shown in Plates 5, 6, and 7, will help make clear the following description:

As previously stated, the range is roughly divided by Tombstone Canyon and Mule Gulch into two parts. To the northeast, and forming about two-thirds of the range, Cretaceous sediments cover the whole area, dipping gently to the northeast and north, and being very little faulted. Where cut by the above-mentioned canyons, they overlie a nearly level plain of schist and granite. This area will be alluded to as the Cretaceous Tract.

The area southwest of the two canyons is almost entirely made up of much faulted and intruded schists and Paleozoic sediments, covered at the southeastern end by Cretaceous sediments overlying a rough topography of Paleozoic beds. It is this area which is studied in detail in this report.

Referring to the accompanying sketch map (Plate 1) of the range, it is seen that in a general way the dips of the Paleozoic beds radiate around the Juniper Flat granite mass. At the northwestern end they dip to the north and northwest. At Mt. Martin and Mt. Ballard, the dip swings to the west and southwest. From Mt. Martin to the southeast, the dips change from south to due east. Each change in dip and strike, however, is generally marked by a fault, the beds seeming to withstand folding to a remarkable degree.

In our division of this area into fault blocks, the divisions given by Ransome[7] have been generally followed, with a few minor exceptions and additions.

Starting at the north is the Bisbee Extension Block with beds dipping to the northwest, and disappearing finally to the north under Quarter-

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[7] *Loc. cit.*, p. 93.

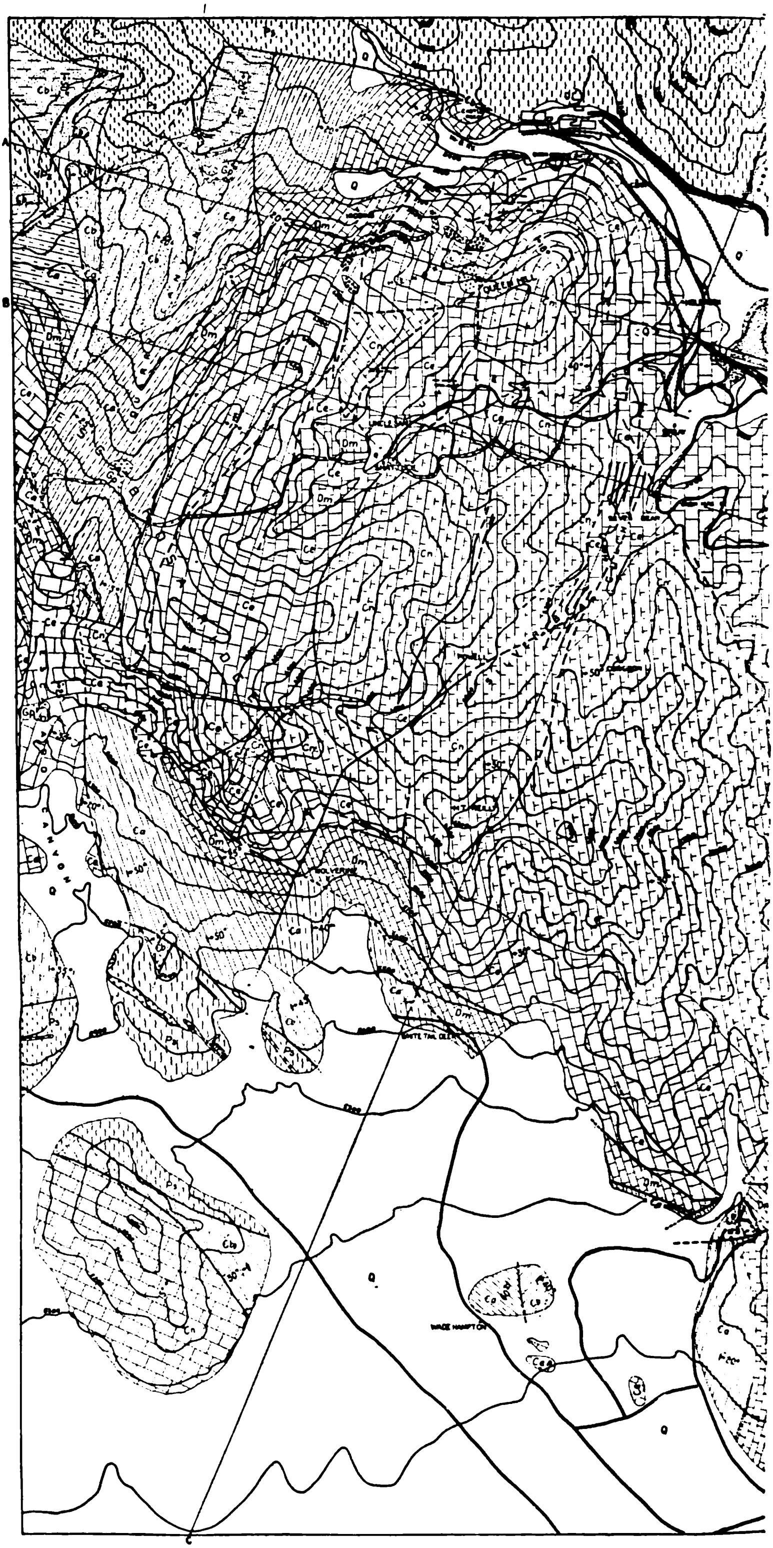
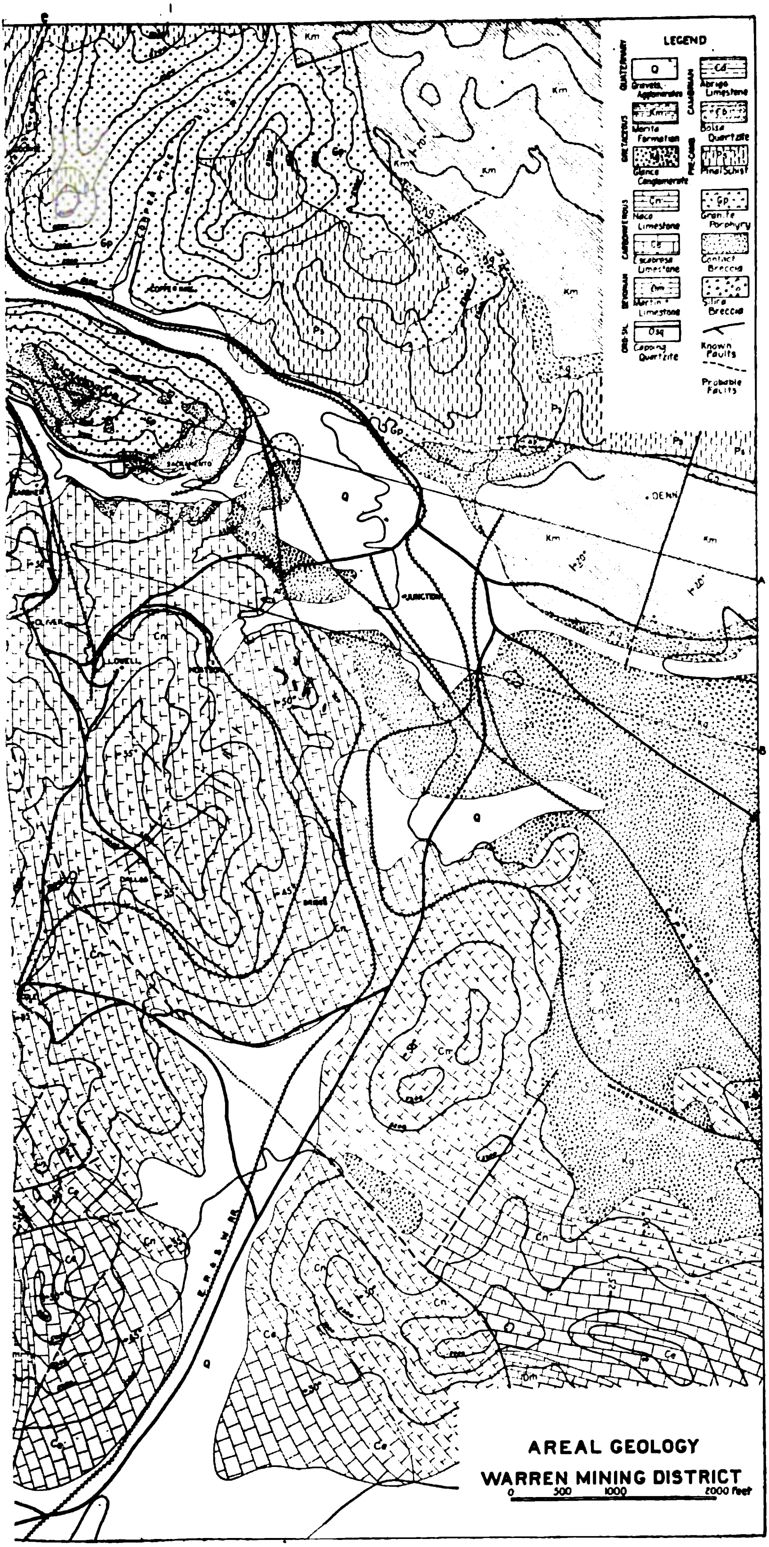


PLATE 4.—AREAL GEOLOGY MAP WITH SECTION LINES A–A, B–B



AND C–C RELATING TO THE SECTIONS SHOWN BY PLATES 5 TO 7.

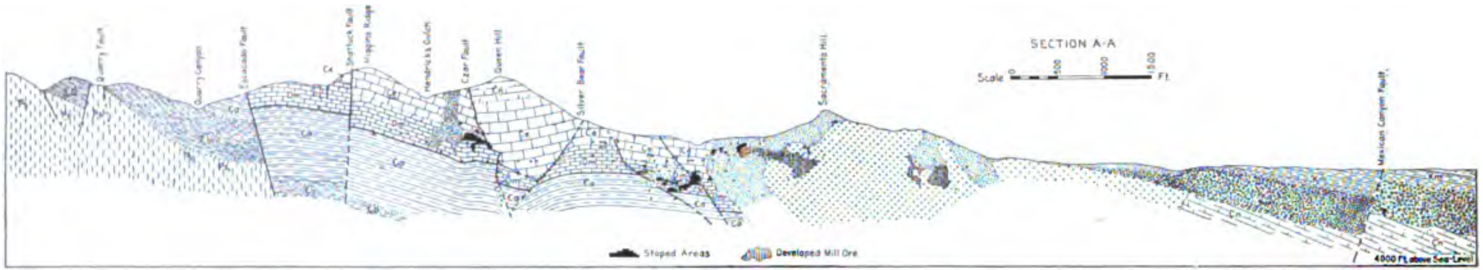


PLATE 5.

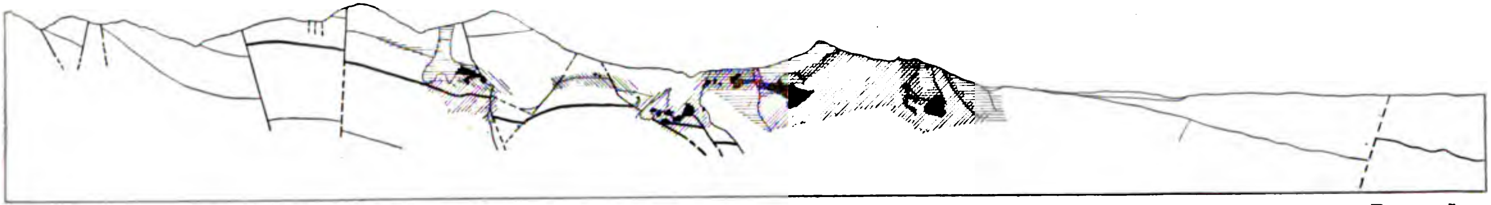


PLATE 5a.

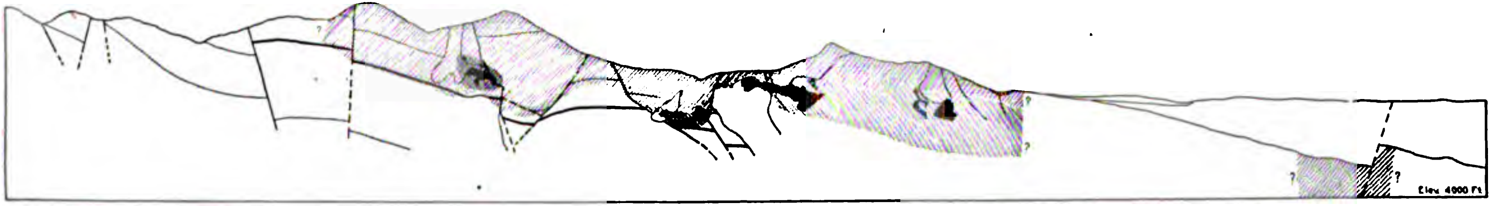

Elev. 4000 Ft.
?
SECTION A-A ZONES DUE TO OXIDATION
Leaching and Limonite Formation
Residual Ores
Sulphide Enrichment
Primary

PLATE 5b.

PLATES 5, 5a, 5b.—GEOLOGICAL SECTION ALONG LINE A–A OF MAP, PLATE 4.

nary wash. This block is bounded on the south by the Modern stockwork of granite porphyry, cutting through the schist and Paleozoic beds and striking northeast and southwest.

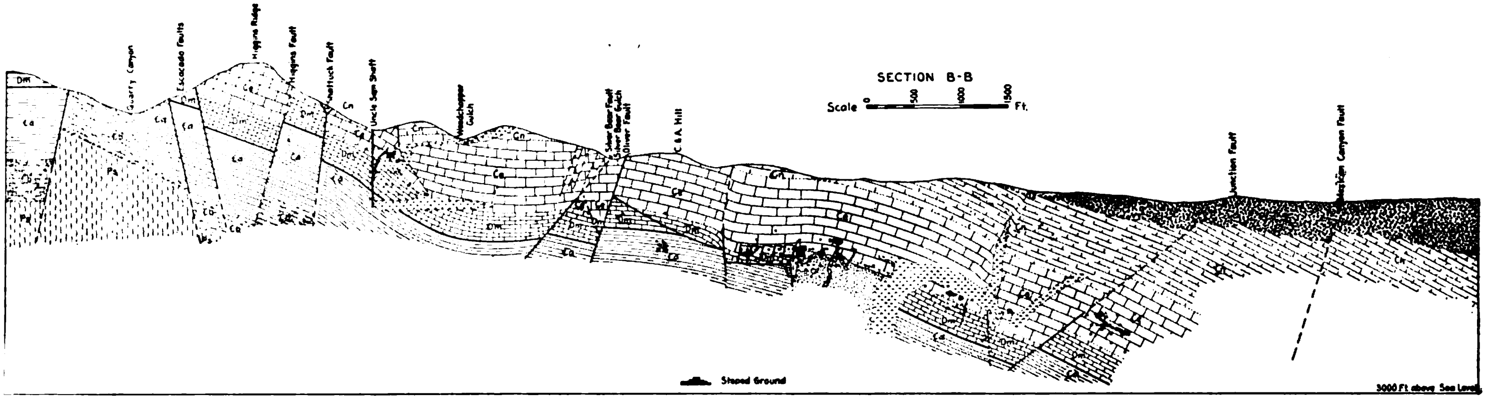
To the south of the Bisbee Extension Block are two blocks, the Escabrosa, extending from the Juniper Flat granite mass to the Escabrosa Ridge line of faults and porphyry intrusions, and the Bisbee West Block extending from Escabrosa Ridge to the southwest. The Escabrosa Block is up-thrown with respect to the Bisbee West Block, and consists almost entirely of schist and porphyry, until Mt. Martin is reached. Here we have the whole Paleozoic section exposed up to the middle of the Escabrosa limestone. The dips at Mt. Martin are southwest. The Bisbee West Block is covered at the northwest by Carboniferous beds dipping to the northwest and west. These are followed to the south by Cambrian beds, dipping to the west and southwest. The Escabrosa Block is bounded on the southeast by the Quarry Fault, beyond which are the Copper Queen and Don Luis Blocks. The Bisbee West Block is bounded on the south by the Abrigo Fault, to the south of which is the Naco Block; relatively very much down-thrown. To the northeast it is bounded by the Bisbee West Fault, to the north of which is the relatively up-thrown Don Luis Block.

The Copper Queen Block is down-thrown with respect to all the surrounding blocks. It is bounded on the north by the east and west Dividend Fault, which has a throw of at least 1,500 ft., throwing the Pinal Schist to the north against Paleozoic beds to the south. It forms a block with the Quarry Fault, a northeast to southwest fault which is the western boundary of the block. The Quarry Fault in turn, forms a block with the White-Tail Deer Fault which is nearly parallel to the Dividend Fault and throws Cambrian beds and Pinal Schist to the south against Devonian and Carboniferous beds to the north. The block is finally covered to the east by Cretaceous beds which are in turn covered by Quaternary wash. To the southeast, the block ends abruptly against the Gold Hill Overthrust Fault. The dips in the Copper Queen Block are generally to the east and northeast.

The Don Luis Block is a small up-thrown block between the Copper Queen and Bisbee West Blocks. The larger part of this block is covered by Quaternary wash, with numerous outcrops of schist and Cambrian beds. The dips in this block are variable but generally to the eastward.

In addition to these are four other blocks, three of which are evidently post-Cretaceous, and one probably so. These are the Tombstone Overthrust Block at the northwestern end of the range thrusting Naco limestone over Morita sandstone; the Naco Block bounded on the north by the Abrigo Fault; the Gold Hill Overthrust Block; and the Glance Overthrust Blocks at the southeastern end of the range.

The Cretaceous Tract, except where the sediments overlie the pre-



Legend As In Plate IX

PLATE 6.

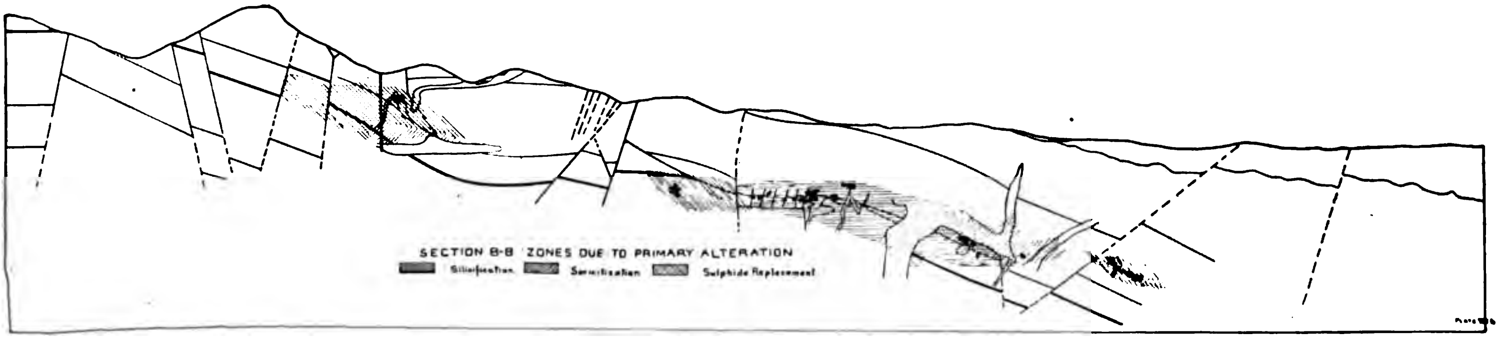
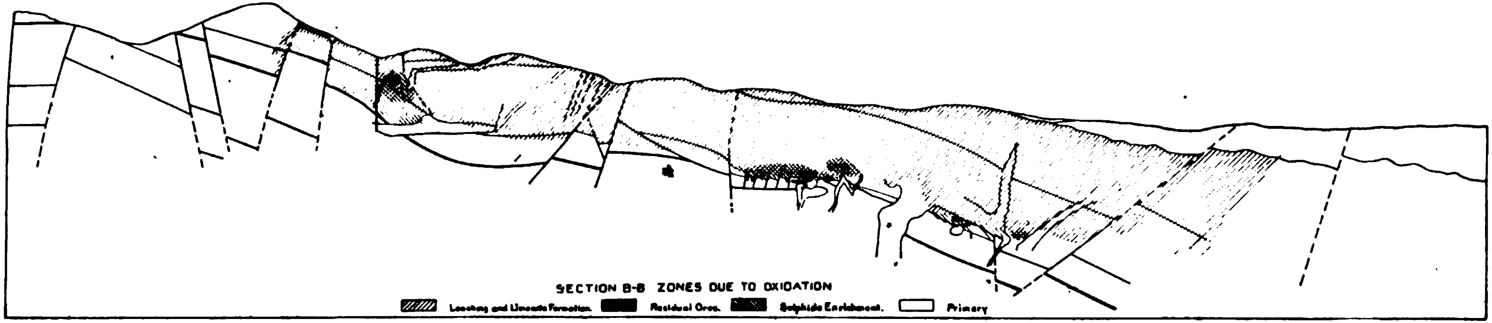


PLATE 6a.


SECTION B-B ZONES DUE TO OXIDATION
▨ Leaching and Limonite Formation.
■ Residual Ores.
▩ Sulphide Enrichment.
☐ Primary

PLATES 6, 6*a*, 6*b*.—GEOLOGICAL SECTION ALONG LINE B–B OF MAP, PLATE 4.

PLATE 6*b*.

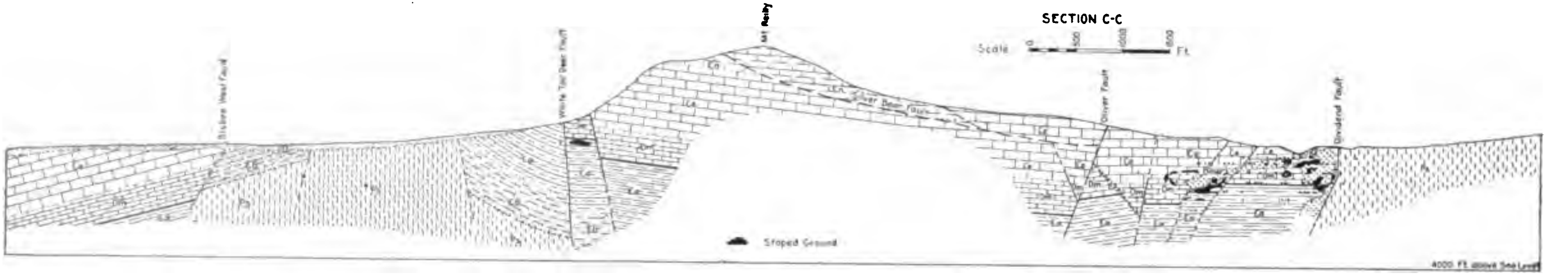


PLATE 7.

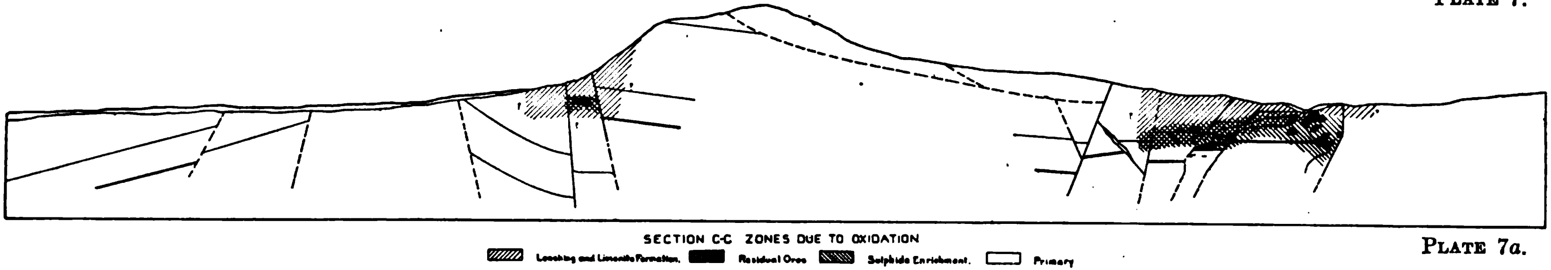

SECTION C-C ZONES DUE TO OXIDATION
▨ Leaching and Limonite Formation.
■ Residual Ores
▨ Sulphide Enrichment.
☐ Primary

PLATE 7a.

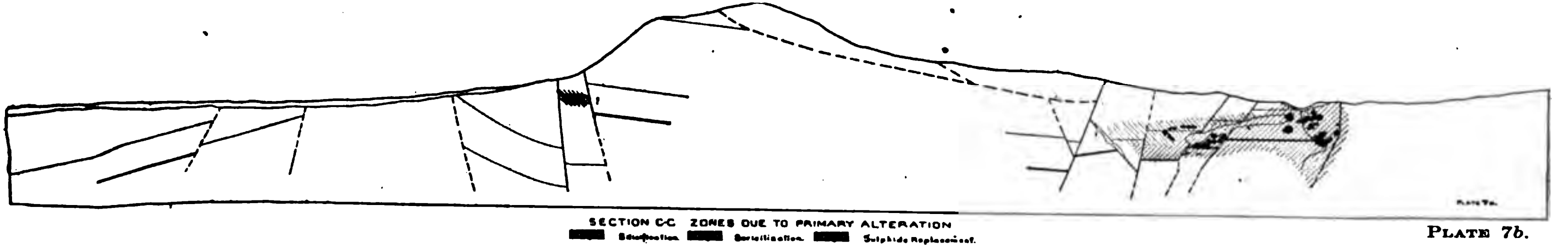


PLATE 7b.

PLATES 7, 7a, 7b.—GEOLOGICAL SECTION ALONG LINE C–C OF MAP, PLATE 4.

Cambrian granite mass, and the Sacramento Hill porphyry, is devoid of igneous rocks. No intrusion has been observed cutting the Cretaceous over this area.

Igneous Rocks

The pre-Cambrian Juniper Flat granite mass has the form of a batholith. The contacts with the schist are clean cut. It probably forced its way up very slowly along some ancient line of weakness. There is no evidence of its having engulfed the schist, as no horses of schist are found inside the area exposed. From the nature of the schist, however, it is difficult to tell how it came in.

In all the pre-Cretaceous blocks mentioned above, extensive intrusion has taken place of granite porphyry, followed in some instances by extrusive rhyolites. These intrusions and extrusions came in for the most part along faults already in existence, but spread out in the less resistant sediments as sills and irregular masses. Although coming in at more or less the same time, some dikes clearly cut the others. The general habit of the porphyry was to follow some fault for a certain distance, and then cut across to another fault. The whole process suggests a very rapid intrusion.

In the Bisbee Extension Block, at the northwestern end of Escabrosa Ridge, there is an occurrence of extrusive rhyolite overlying an erosion surface of Naco limestone and an intrusive porphry sill.

Cutting through the Sacramento Hill porphyry, north of Naco Road, are several small dikes of andesite. This same andesite occurs cutting Escabrosa limestones near the Cole shaft, and has been observed underground near the Shattuck, Wolverine and Wade-Hampton mines. At the Shattuck mine, it is undoubtedly later than the porphyry. This andesite in all observed cases has come in as very small dikes in already opened fissures. In only one case has an outcrop been observed in higher beds than the Martin.

In the post-Cretaceous blocks, no intrusions have been seen except in the Naco Block. Here, at the north end of the block, entering along the big Abrigo Fault, an extrusive rhyolite mass has come in. This mass is similar in all respects to the mass mentioned above in the Bisbee Extension Block. Except for this one case, the large faults of post-Cretaceous age have not been accompanied by igneous intrusion.

The most important, economically, of all these igneous rocks is the granite porphyry, as it alone seems to play an important rôle in the processes of ore deposition. By far the greater amount of porphyry in the district cuts through with clean contacts. The one exception to this rule is the mass of Sacramento Hill. The mode of intrusion of this particular mass is somewhat different from all the others, due to the accompaniment of mineralizing emanations. This will be more fully treated later.

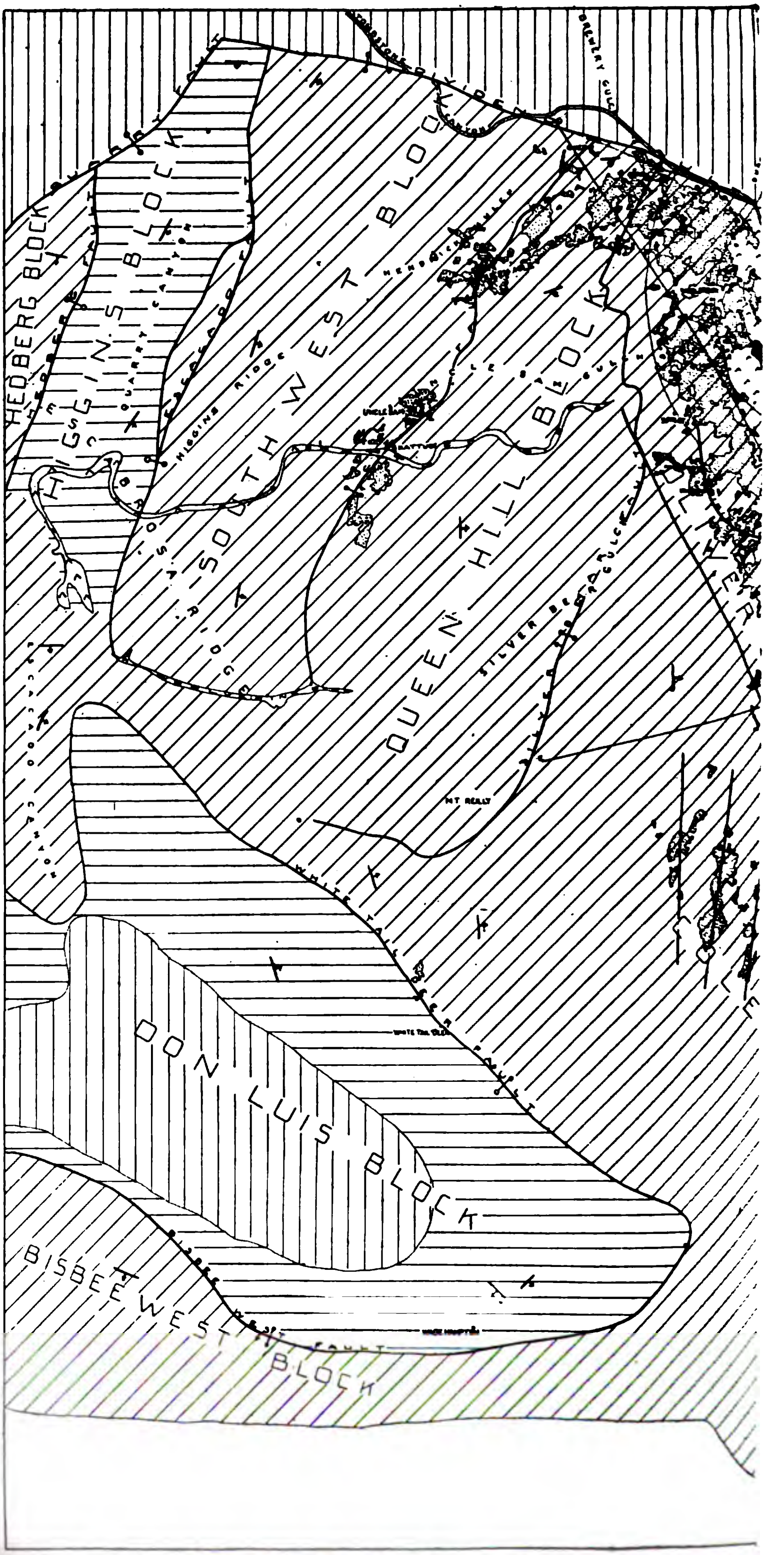
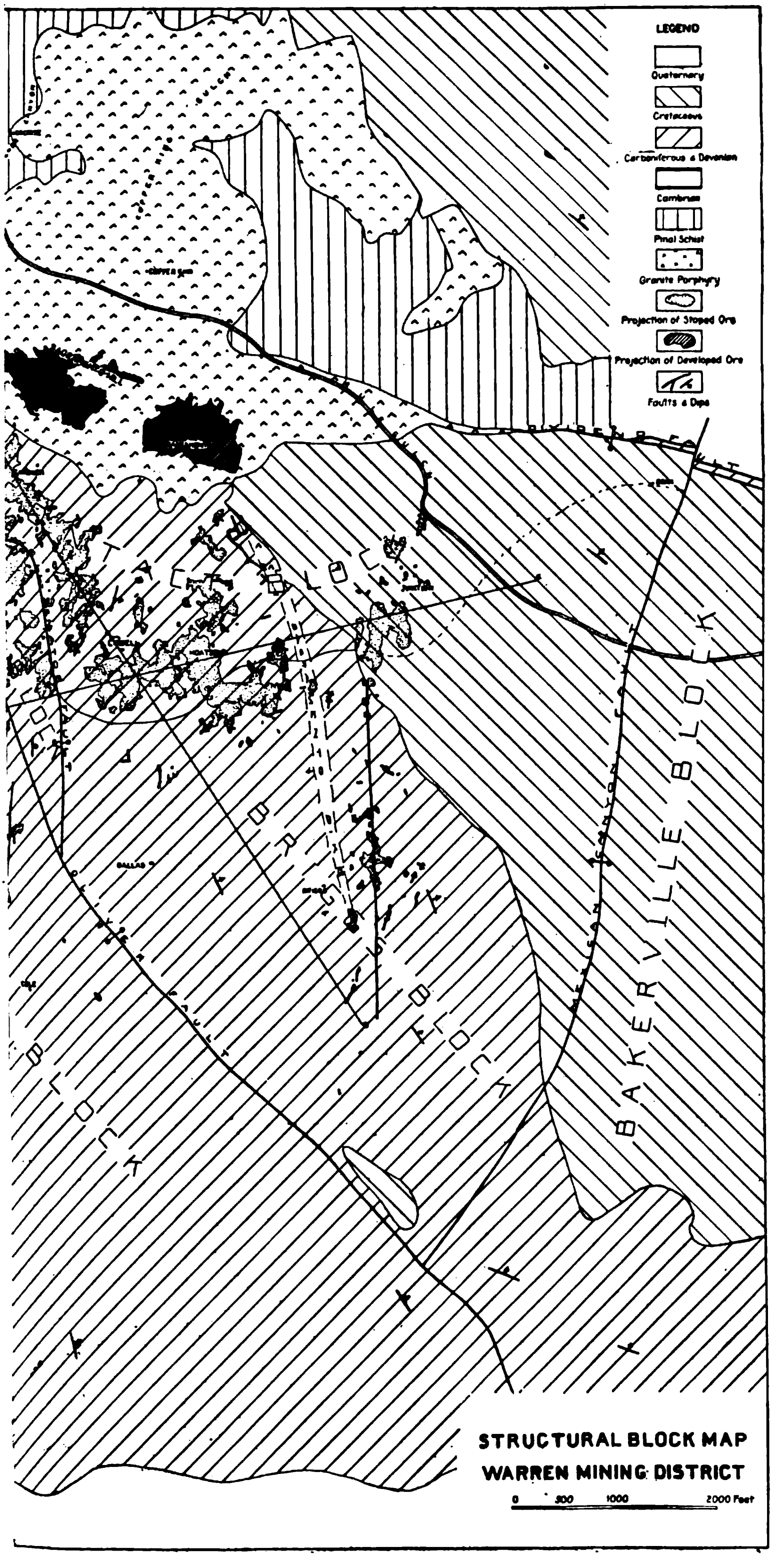


PLATE 8.—(SAME AREA AS PLATE 4) SHOWING HORIZONTAL PROJECTION



OF STOPES AND SECTION LINES D–D AND E–E OF PLATES 9 AND 10.

COPPER QUEEN BLOCK

Fault Blocks

From an economic viewpoint, the Copper Queen and Don Luis Blocks are at the present time the only important ones in the range, and these two blocks, together with the mode of intrusion of the Sacramento Hill porphyry mass, will be more fully explained below.

The Copper Queen Block, whose details are best shown by Plate 8, lies in the heart of the range. It is a relatively down-thrown block, being bounded on the north by the Dividend Fault, a N. 75° W. fault dipping to the south from 60° to vertical. This fault has an estimated throw at the town of Bisbee of at least 1,500 ft. throwing Pinal Schist with a few outliers of Bolsa Quartzite on the north against Paleozoic sediments to the south. This fault apparently forms a block to the west with the Quarry Fault, a N.E.-S.W. fault which forms the western boundary of the area. To the south, the block is bounded by the White-Tail Deer Fault system parallel to the Dividend, throwing Bolsa quartzite and schist to the south against Abrigo and Martin limestones, to the north. To the east, the block is covered by Glance conglomerate and Morita sandstone, which are in turn covered by the Quaternary gravels of Sulphur Springs Valley. In the southeast, the block ends abruptly at the Gold Hill Overthrust Fault.

Starting somewhere near the Spray shaft, the block is cut by a N.W.-S.E. fault of considerable throw, the Oliver Fault, which dips about 60° S.W. This fault increases in magnitude toward the southeast and where it is finally covered by Quarternary wash, about halfway between Don Luis and the Country Club, it throws Abrigo limestone to the northeast against Naco limestone to the southwest, a throw of at least 1,500 ft. This fault is encountered in the Spray, Oliver and Cole workings. At the Spray the throw is about 200 ft.; at the Oliver it throws Abrigo limestone against Escabrosa with a throw of at least 500 ft. It finds topographic expression in the big draw between the Dallas and Cole shafts.

The Copper Queen Block, as seen by Plate 8, may be roughly divided into eight structural blocks. Starting from the west, we have first the Hedberg Block, relatively down-thrown, and with dips generally to the south. This is followed by the Higgins Block, up-thrown with dips to the east and southeast. Following this is the Southwest block, bounded on the east by the Czar fault system, and on the west by the Escacado fault system. This block is down-thrown relative to the Higgins Block but up-thrown in relation to the Queen Hill Block to the east. The dips here are nearly due east. The Queen Hill Block is a funnel-shaped block of Naco and Escabrosa limestone. It is well shown in section A-A, Plate 5. It is bounded on the west by the Czar fault system, and on the east by the Silver Bear Fault. The dips on the surface are irregular but, in general, to the east. Underground, below where the two faults wedge out, the dips are consistently east. Surrounding Sacramento Hill, to the south and southwest is a dropped block, which we may call the Contact Block. It is bounded on the north and northeast by the Dividend Fault and Sacramento Hill, and on the south and southwest by the Dixie-Howell-Senator Faults. To the southwest of this Contact Block and up to the Oliver Fault is the Oliver Block of relatively up-thrown beds. Southwest of the Oliver Fault is the Cole Block and northeast of the same fault is the Briggs Block which extends east to the post-Cretaceous Mexican Canyon Fault. The dips of all these blocks more or less surrounding Sacramento Hill are nearly due east. The Oliver Fault has tended, however, to swing the up-thrown side, especially where the throw becomes greatest, around to the northeast as is shown in the detailed geological map of the district, Plate 4.

To the east of all these blocks is an eighth block, the Bakerville Block, almost entirely covered by Glance Conglomerate. This block is bounded on the west by the Mexican Canyon Fault, striking about N. 15° to 30° east and dipping to the northwest. This fault is a late one and extends beyond the Dividend Fault, causing the prominent dislocations of the Mural Limestone seen north of Lowell and Warren.

As is shown in section B-B, Plate 6, the apparent dip of the beds in the whole block is very much less than the actual. This is due to minor faulting or slipping which has had the effect of dropping the beds very consistently to the west and raising them to the east. This is one of the typical structural features of the camp and seems to be caused by the extreme resistance of the beds to folding.

There are two ages of faulting, one pre-Cretaceous and the other post-Cretaceous. As has previously been shown, the porphyry in the district is pre-Cretaceous. The age of the faulting has therefore been determined in large part relative to the age of intrusion. The only large fault which is definitely post-Cretaceous is the Mexican Canyon Fault. As the fault apparently blocks with the Oliver Fault as shown in Plate 8, the Oliver is probably also post-Cretaceous. This is borne out by the character of the exposures underground. There, practically no mineralization accompanies it, and in Oliver ground it apparently cuts off an orebody, which as will be seen later, would make it post-Cretaceous. Both of these faults have the same effect, that is to raise the sediments to the east, and to drop them to the west. The probability is that the Junction Fault, shown in section B-B, Plate 6, is also post-Cretaceous, as it is reported to cut off certain parts of the Junction orebodies, and it is also possible that a large part of the north to south slipping in Lowell and Gardner ground is also post-Cretaceous. Here the throw of each individual fault is so slight, and the ground so highly metamorphosed, that it is not susceptible of proof one way or another.

All other faults, so far as observed, have had their major movements in pre-Cretaceous time and before the introduction of the porphyry. Some faults have had movements in both ages. The Dividend shows some movement at the present time, and enough movement occurred in post-Cretaceous time to throw out the Glance Conglomerate. Some post-intrusion slipping has also undoubtedly taken place along the Czar Fault.

Intrusions

As has previously been shown, the block has been extensively intruded by granite porphyry. The largest and most important mass is that of Sacramento Hill, the other occurrences being more or less radial offshoots of dikes and sills from this central mass. It has also been shown that underground exposures below the Escabrosa horizon are much greater than surface exposures.

The path of the Sacramento Hill porphyry was the previously opened-up Dividend Fault. Underground development has shown that the porphyry acted as a plug, and that none of the later movements on the fault have taken place along the original line through the mass. The offshoots of the mass have generally followed major lines of weakness such as the Czar, and Silver Bear Faults. Some, however, such as the Shattuck dike, have cut across the country irrespective of fault lines.

The Sacramento Hill, and the connecting Lowell, mass differ from all the other porphyry occurrences in the range, in that they were accompanied by mineralizing emanations, which have completely altered them. These emanations, besides altering the porphyry itself, changed and broke up the surrounding sediments. In consequence of this, as the mass worked its way up, it dragged in pieces of schist, quartzite and limestone along its edges. The final result was a central core of metamorphosed porphyry, with a periphery of contact breccia, made up of highly silicified and altered fragments, sometimes rounded and sometimes angular, of schist, porphyry, quartzite and metamorphic limestones. This contact-breccia zone has a variable width, with a maximum of about 1,000 ft. It has fairly sharp contacts with the surrounding limestone, but tends to grade off into brecciated porphyry, and finally blocky unbroken porphyry, toward the center of the intrusive mass.

DON LUIS BLOCK

The Don Luis Block is an elliptical-shaped block up-thrown relative to the Copper Queen. It is bounded on the north and east by the White-Tail Deer fault system, throwing Abrigo and Martin limestone against schist, Bolsa, quartzite, and some Abrigo limestone. To the south it is bounded by the Bisbee West Fault, which is covered by the Quaternary wash toward the east and toward the west it is bounded by the Quarry Fault. The dips are variable, but for the most part are to the northeast and east.

Intrusions and Orebodies

The block is intruded both by granite porphyry and by andesite, the latter, however, not outcropping. There are two ore occurrences in the block, one at the northern boundary, and rightfully in the Copper Queen Block, and one at the southeastern end. The first of these, the White-Tail Deer orebody, occurs as a replacement of Abrigo limestone, about 50 ft. from the base of the Martin. It is not associated with any known intrusion, but is close to the White-Tail Deer fault system. The second occurrence is that of the Wade-Hampton. Here the ore occurs as lead and copper ore in a fault separating Abrigo limestone from Bolsa quartzite. The ore here is apparently associated with a dike of andesite and occurs in the crushed Abrigo limestone on the fault.

VI. GEOLOGIC HISTORY

The earliest geologic age is recorded in the district by the pre-Cambrian Pinal Schist. As has been previously stated, this was probably derived from arenaceous sediments.

Due to the extreme metamorphism, the geologic history subsequent to the deposition of these early sediments, and up to the deposition of the Cambrian sediments, is obscure. The sediments were probably subjected to extreme folding, intruded by diabase, and deeply buried. After the metamorphism was complete, the schist was intruded by a large mass of granite. From then on to Cambrian time, erosion worked on the whole complex, wearing it down to an even peneplain.

At the beginning of Cambrian time, a gentle subsidence took place, resulting in shore-line deposits of quartzites, the subsidence becoming more rapid toward the end of the period, in which time were deposited finer sands and some shales. The land surface toward the end of the Cambrian period became more distant, as evidenced by the deposition of the Abrigo shales, and shaly limestones.

At the end of Cambrian time, a gentle rise took place over an extensive area, bringing the flat sea bottom close to the surface, with land actually emerging probably toward the south. During the Ordovician and Silurian ages, this lowland surface of Cambrian beds was subjected to slow erosion simultaneously with the deposition in the district of an 8-ft. bed of quartzite. A slow subsidence then took place to the north of the district, allowing the thicker deposits of quartzite in the Yellowstone Range of 40 ft., and at Globe of 500 to 700 ft.

At the end of the Silurian, a sudden subsidence took place, resulting in deeper-sea conditions and thus the Devonian age is represented by fairly pure dolomitic limestones.

The sea evidently continued to deepen all through Mississippian time with the deposition of the pure Escabrosa limestone. During Pennsylvanian time, subsidence did not keep pace with deposition and the sediments became increasingly more shaly, and cherty showing a nearer approach of land surface and shallower seas. At the end of the Pennsylvanian, and probably during the Permian age, a violent uplift took place, resulting in the doming of the sediments around the pre-Cambrian granite mass. The sediments were shattered by extensive faults, the major lines of weakness being northwest and southeast. The granite mass itself resisted this shattering to a great degree except around its edges. Immediately following the faulting, or partly accompanying it, granite porphyry was intruded, following generally the major lines of weakness, but also shooting sills and dikes into the less resistant sediments. Accompanying one of these porphyry intrusions, that of Sacramento Hill, were mineralizing emanations, which metamorphosed the surrounding sediments and the porphyry itself, and were the source of our present orebodies.

During Jurassic and Triassic time, the whole area was subjected to erosion, during which time most of the upper Carboniferous beds were stripped off, and at the crest of the dome, around the granite and schist of Juniper Flat, the old pre-Cambrian complex was laid bare. During this erosional period, the orebodies were subjected to oxidation, which would place the age of primary ore formation definitely at or near the age of the uplift.

At the end of this period, a remarkably sudden subsidence took place, before any of the land except the top of the dome had been leveled off, and the rough topography surrounding was filled up with the extremely coarse Glance Conglomerate. The subsidence was so sudden that the fragments making up this conglomerate were hardly worked over at all, it being made up of angular boulders, some as large as a yard in diameter, of schist, quartzite and limestone. The roughness of this old topography is well shown in the basin covered by conglomerate, in which the town of Warren is situated.

After the whole country was leveled off by the Glance Conglomerate, the subsidence became less rapid, and shore-line deposits of sandstones and sandy shales were laid down, followed by deeper-sea conditions, during which the pure Mural Limestone was deposited, showing abundant Cretaceous (Comanche) fossils. Shore-line deposits followed this again to the end of Cretaceous time.

During Tertiary time, a second uplift took place, along almost the same axis as the previous one. This also took the form of a dome around the old granite mass, and the Escabrosa Ridge porphyry intrusion. This uplift, however, was not as violent. It was accompanied by extensive block faulting, with little disturbances of the beds within the blocks. The major faults took the form of overthrusts, the largest being the Tombstone and Gold Hill faults. Either accompanying the uplift or subsequent to it, some intrusion of monzonite porphyry took place, at the southern end of the range. Probably some of the extrusive rhyolite also belongs to this age of intrusion. Finally the whole range was subjected to a tilt to the northeast of about 15°.

From then on to the present time, erosion has been steadily at work, and has again laid bare the old pre-Cretaceous sediments along the crest of the dome, with Cretaceous beds exposed to the northeast. To the southwest, the Cretaceous has been all eroded, due to the final tilting. The only evidence left of the second doming is found at the southern end of the range, as shown by Plate 1. During this last erosion period the orebodies have been further subjected to oxidizing and enriching processes, so that we owe their condition as we find them today, to two widely separated periods, in each of which the original primary ores were worked over by surface waters.

VII. METAMORPHISM AND MINERALIZATION

General Metamorphism

In a region that has been subjected to as many geological changes as the Warren district, it was to be expected that the rocks would show many variations in texture and mineralogy from the unaltered types. The effects of regional metamorphism as well as dynamic are represented in the Pinal Schist as has been shown in describing the lithological characters of this formation.

The Cambrian formations have also been changed throughout their occurrence in the district by a process related to regional metamorphism in the wide scope of its action. The induration of the pure Bolsa quartzite has gone on until it is of glassy texture, but in the clayey, magnesian Abrigo limestone, besides the segregation of chert, the impurities have everywhere combined into such minerals as epidote, zoisite, chlorite, serpentine, and possibly albite. No such action is at all observed in the conformably overlying Martin limestone, which strongly suggests that the changes took place in the long period separating the middle Cambrian from the Devonian. But since field evidence excludes any great disturbance during the unrepresented period, it is probable that the general metamorphism in the Abrigo was caused by the deep burial under more than 4,000 ft. of overlying limestones after the impurities in the beds had had a chance to segregate and rearrange themselves due to the slow action of cold surface solutions during Ordovician-Silurian times. The necessary temperature conditions were probably obtained by the rise in isotherms due to deep burial and to the post-Carboniferous intrusive period when a general rise in temperature could effect changes in the more susceptible impure sediments.

Whether this tentative explanation is correct or not, the importance of recognizing the general distribution in the Abrigo of minerals which may be mistaken for indications of contact metamorphism cannot be overemphasized. This formation is of considerable economic importance and the underground search for ores cannot be aided by the same indications as in the higher limestones.

Contact and Hydrothermal Metamorphism

That the granite-porphyry intrusion was the supreme factor in the ore deposition of the district cannot be doubted in view of the intimate association of the ores with the porphyry and the intense changes that have taken place in the rocks around the main intrusions.

It has been customary to separate the metamorphism of the rocks around a contact into contact metamorphism, caused by the igneous rock itself, and hydrothermal metamorphism, caused by the heated solutions emanating from the cooling magma. This distinction will not be made in the present article, because there has been nothing found by the authors to indicate that there is any genetic difference in the causes for changes in the rocks, outside of differences in intensity or quantity. The differences in *effects* are not to be confused with the causes, as the first are subject to the many variations of the rocks encountered, their physical and chemical differences, and their previous alteration. The metamorphosing agencies vary also with the distance from their source in the effect they may have on the same rock, but all changes observed so far have usually been graded and not susceptible to sharp definition.

In general, it may be stated that the two main centers of porphyritic intrusion are also the centers of metamorphism. Around Sacramento Hill the effects of metamorphism can be seen on the surface, but around the Lowell center the alteration did not extend through the covering of upper Carboniferous limestone. The effects around Sacramento Hill are more pronounced on the side of the calcareous sediments, which have been so much more altered than the schist of the north side. The porphyry on the south, as well as the covering remnants of contact breccia, are highly silicified and stand out as the crest of Sacramento Hill. The topographical depression around this hill is due to less silicified porphyry breaking through the contact breccia where this last turns from a more or less vertical to a horizontal body. Outside of this moat-like depression is a ring formed by the highly silicified contact breccia which grades into silicified metamorphic limestone, and this into limestone with a decreasing amount of metamorphic minerals until a belt of marbleized rock is reached, from 200 to 1,000 ft. away from the porphyry. The zone of recrystallization also fades outward irregularly to the unaltered sediments.

On the north, or schist side of the contact, there is some silicification of the older rock, accompanied by considerable pyritization, but this very soon fades out into the unaltered schist.

Underground, in tracing out the zones of rock alteration due to the intrusion and its accompanying solutions, it has been found that they roughly correspond to the ones just mentioned as appearing on surface, with the following general modifications: The zones extend farther than the general surface contours along extensions of the porphyry mass, especially the zone in Lowell ground, and along lines of fracturing which may or may not also be lines of faulting. The extent of the zones is remarkably influenced by the formation they are traced in, the changes going farthest at the top of the Devonian and the top of the Cambrian. Local variations are found to break the arrangement of the zones of alteration where there are minor centers of strong action, such as close to the Shattuck dike in the Uncle Sam and Shattuck ground, in the Southwest mine, and in several other places.

The Effect of Contact and Hydrothermal Metamorphism on the Formations

Before taking up the effects of contact and hydrothermal alteration on the rocks of the district it is well to state that hydrous metamorphism or oxidation has obscured or complicated in a great measure the results of contact metamorphism, and in many cases made the exact separation of the process involved too difficult and useless to be undertaken in the course of economic work.

The changes due to circulation of meteoric waters will be taken up later. The mine workings have now gone down far enough in all zones to disclose the fact that there is a depth below which the rocks as well as the ores are different from those closer to the surface. Here there are none of the minerals evidently due to the process of oxidation, and mineralogical associations continue the same indefinitely downward, without the vertical variations of the upper portions. This is the criterion that has been used in separating the effects of primary and secondary processes in the rocks and in the ores. Though not found in this order in the mines, primary effects will be considered first.

As a whole the impression given by the rock alterations in the Warren district is one of abundance and persistence rather than great intensity. There are no high temperature minerals developed in any great amount, or if they were developed at some stage, they have now been replaced. Garnet, diopside, wollastonite, scapolite, and vesuvianite have been observed, but in very small amount, and never forming an important part of a formation. Tremolite, actinolite, and edenite are far more common.

The distinguishing minerals of alteration zones are quartz, sericite, chlorites, especially penninite, and the oxides of iron, magnetite and specularite, as well as the metallic sulphides of iron, copper, zinc and lead.

Metamorphism in Contact Breccia

Around any of the granite porphyry that has been accompanied by mineralizing emanations, such as that of Sacramento Hill, the contact breccia is marked generally by an extreme amount of silica which replaces all other gangue and rock-forming minerals. Pyrite and sericite are next in abundance, with chloritic minerals in variable amounts and calcite practically unknown. Schist and quartzite fragments have naturally remained the least altered in this mass and are consequently recognized most easily. In this contact breccia, or at the edges, there are bodies of intergrown magnetite, hematite and pyrite, with associations of the best-formed garnet, wollastonite, and other contact minerals. Silicification usually decreases toward the outer edge of the breccia, sericite and chlorites increasing. In many thin sections studied, the prevailing impression gathered is that in most of the breccia silicification is the last process involved, following sericitization and chloritization.

There are, however, distinctly later porphyry dikes cutting the breccia, as has already been stated, which have also been accompanied by metamorphosing emanations, the result being a complication and confusion of the processes involved. Certain highly chloritized portions of the breccia, where penninite replaces quartz in a noticeable way, are the result of these conditions, which are economically important, since copper minerals are concentrated in very pure form by the superimposed alteration. There are also other observed forms of this repeated alteration depending on the variations of either the first or the last. But it is generally true that the second process never reaches the point of adding silica or even adding sericite to the previously changed rocks.

Metamorphism in the Contact Limestones

The transition from breccia to limestone that can be recognized is apt to be very sudden in the Carboniferous horizons, and graded in those below. This is because the alteration of the lower limestones is more pronounced. The impure, and at the same time easily crushed, Devonian and Cambrian formations become a mass of sericite, penninite, calcite, and quartz, with some tremolite, garnet, diopside, wollastonite, and epidote, while the Escabrosa limestone usually has just a narrow fringe or a few bands of metamorphic minerals and the mass of the rock is simply recrystallized to marble. As extremes of alteration in this contact zone there are places where the limestone, especially the Devonian, is converted into a highly sericitic or chloritic mass, or into almost pure quartz.

There is some evidence in this zone also that silicification is the last process affecting the rocks under ordinary conditions.

Farther from the contacts or the centers of mineralization the changes visible in the limestone are a decrease in silica and sericite in all the rock, while these minerals may persist in breaks and joints. Recrystallization of calcite, but not of the impurities, is very common.

In this contact zone the accumulation of sulphides is marked by the increase in either sericite, silica, or chloritic minerals around the sulphides.

Metamorphism Outside of Contact Zones

Sometimes rocks with considerable alteration are encountered far from any known porphyry, in which case it may be that solutions have traveled along fractures, or that they may have come with undiscovered porphyry. Usually, however, the copper ores have been found replacing the limestone very much farther away than any general alteration of the formations, in which case the transition from fairly unaltered rock to sulphides is sudden, but bears some relation to the local structure.

Metamorphism of the Porphyry

The primary solutions emanating from the granite porphyry affect that porphyry in about the same way as they do the adjacent rocks. If the nearby rocks, whatever they may be, are silicified, the porphyry will be also. If sericitized or chloritized, the same processes will have affected the igneous rock. The main difference comes in the relative amount of sulphides which have formed during these changes. Of course, the same minerals will not form in limestone as in igneous rock, but an increase or decrease in silica or magnesia or iron will be parallel in both cases.

The main mass of Sacramento Hill is so highly altered that a thin section of this rock shows only a mat of sericite plates, with variable amounts of quartz, pyrite, and chlorite, without much being left to distinguish between quartz, feldspar and mica phenocrysts, and the base. Close to the breccia, silicification is common, accompanied by disseminated copper deposits.

In the dikes radiating from the hill, the changes in alteration of the porphyry are very clearly marked, as well as the relative ages of succession of the processes. Receding from the silicified portions, sericite increases and the outlines of phenocrysts, first of quartz, then of feldspar, begin to appear. Lastly appear the outlines of biotite, marked generally by successive laminæ of sericite and pyrite. Farther away penninite is found in increasing amount, the quartz phenocrysts being unattacked and some of the plagioclase feldspars scarcely altered. The chloritic minerals appear as remnants replaced by sericite.

When sericite has almost all disappeared, and there is no silicification, epidote, zoisite and calcite are sometimes found as alteration or metamorphic minerals in the porphyry. At this zone there are also found portions of the rock converted into almost pure penninite and edenite, with some serpentine, as if the ferromagnesian minerals lacking in the more sericitic rock had been driven to accumulate and replace other portions. Only the presence of some unaltered quartz phenocrysts and very resistant apatite crystals helps to determine the original rock, although transitions have been exposed in some of the workings.

Sulphides in some amount invariably accompany the contacts of porphyry that is altered to this degree.

Farther away from the centers of action slight chloritization of the base, of some feldspars and of micas, and eventually of the micas alone, is the only alteration noticeable. This fades out in a few observed cases to fresh rock, or rock affected only by meteoric waters. The contacts of this less altered porphyry have either just a little pyrite and sphalerite, or no sulphides at all, and even have had no appreciable effect on the immediately adjacent limestone when seen in thin section.

The alteration of the more basic dikes which cut the contact breccia is also similar to the one induced in the breccia, except that the earlier changes are not observed in these dikes.

SUMMARY

To summarize, it may be said that the weakest as well as the farthest reaching alteration in the porphyry is chloritization. Next comes epidotization, which is of minor importance. Sericite and finally quartz are the additional minerals formed with the increase in the strength as well, probably, as the temperature of the emanations. The minerals formed by the stronger solutions are formed by replacement of the earlier ones, in the ordinary case when metamorphism is advancing in intensity from the center. Superimposed alteration may be caused by later intrusives, or in certain instances by the decrease in intensity in the dying-out period of alteration.

Some unaltered dikes in Holbrook and Czar ground are found in zones where the rest of the rocks show intense sericitization, including some masses of porphyry. These dikes are associated also with some sulphide ores, the contacts being far more distinct than ordinarily. If it were not for the slight sericitization of these dikes close to the ore, while away from it they show no hydrothermal action, it would be possible to believe that they are the result of post-mineralization intrusions.

The Shattuck dike shows similar phenomena, as it is very fresh for long stretches, and then, close to the ore zones in Uncle Sam and Shattuck ground is almost as altered as any rock around Sacramento Hill, showing that the altering solutions did not accompany all the porphyry but found their way through special channels.

The following partial analyses of altered porphyries are typical:

	I	II	III
Au, ounces	*Nil*	*Nil*	*Nil*
Ag, ounces	0.04	0.08	0.04
	Per Cent.	Per Cent.	Per Cent.
Cu	0.24	0.11	0.11
Pb	Tr.	Tr.	Tr.
SiO_2	55.9	76.6	73.3
Fe	9.8	5.6	5.7
CaO	1.8	3.1	0.5
MgO	0.5	3.2	2.8
Al_2O_3	16.6	7.4	10.1
S	8.8	1.0	0.4
Mn	0.6	0.6	0.6
Zn	Tr.	Tr.	Tr.
P	0.08	0.10	0.09
Totals	94.32	97.71	93.60

I. Highly sericitized porphyry with pyrite, from Sacramento dike, close to Sacramento Hill.

II. Chloritized, partly sericitized and epidotized porphyry, from Sacramento dike, about 1,000 ft. away from the hill.

III. Slightly chloritized porphyry, Dallas mine.

The general zones of primary alteration are shown for the sections through the camp in Plates 5*A*, 6*A*, and 7*A*. Silicification and sericitization have been explained already. Chloritization is not marked because it is too irregular, and though important in the porphyry, the sediments in that zone are not generally affected. The zone of sulphide replacement was introduced, where the limestones surrounding sulphide replacements were affected so slightly immediately away from the orebodies that there can be said to be practically no metamorphism.

VIII. THE ORES OF THE DISTRICT

General Occurrences

The general occurrence of the orebodies can be best appreciated by reference to the horizontal and vertical projections of the stopes given in Plates 8 to 10.

In the horizontal projection (Plate 8), there is a well-marked crescent-shaped zone around Sacramento Hill, with an extension around the Lowell mass of granite porphyry. Along the Dividend and Czar zones the effects of major structural lines are well marked by the abundance of ore taken out. The Shattuck dike-Czar zone intersection has its group of orebodies, and extensions and outliers from the general zone follow both the Sacramento dike and the series of breaks in Briggs and Cole ground to the southward.

In the vertical projections (Plates 9 and 10) the inclination of the ore zone is shown; also in Plates 5 and 6 the stratigraphical position is seen to follow in general the dip of the limestones, at horizons varying from the top 300 ft. of the Cambrian to the lower 300 ft. of the Carboniferous.

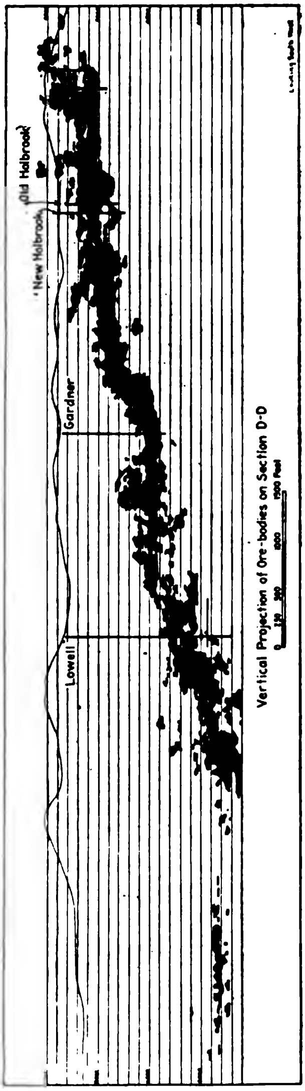


Plate 9, Section D-D.—Showing Inclination of Ore Zone Following Dip of Limestones.

Classification of the Orebodies

As in the case of the metamorphism and alteration of the rocks, the authors have found no reason to separate the orebodies due to genetic differences. All the ores are believed to be due to one general period of primary mineralization which followed closely the intrusion of the main porphyry masses. Differences in the ores are ascribed mainly to the character of the formation that they replace and to the amount of general metamorphism in the surrounding rocks due to the distance from the centers of alteration. All the ores are also believed to be formed by metasomatic replacements of various rocks.

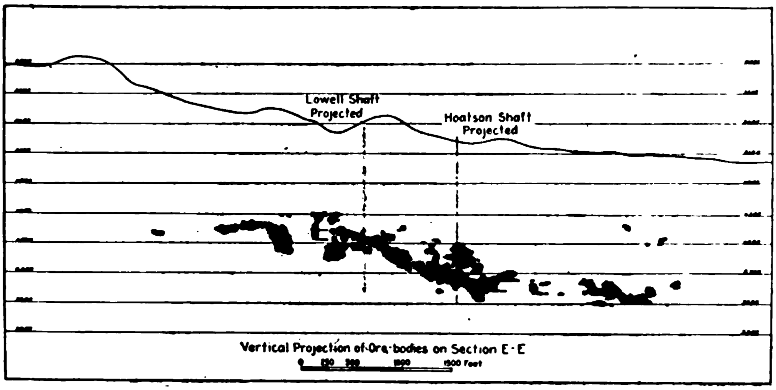


Plate 10, Section E–E.—Ore Zone Between Lowell and Hoatson Shafts Following Dip of Limestone.

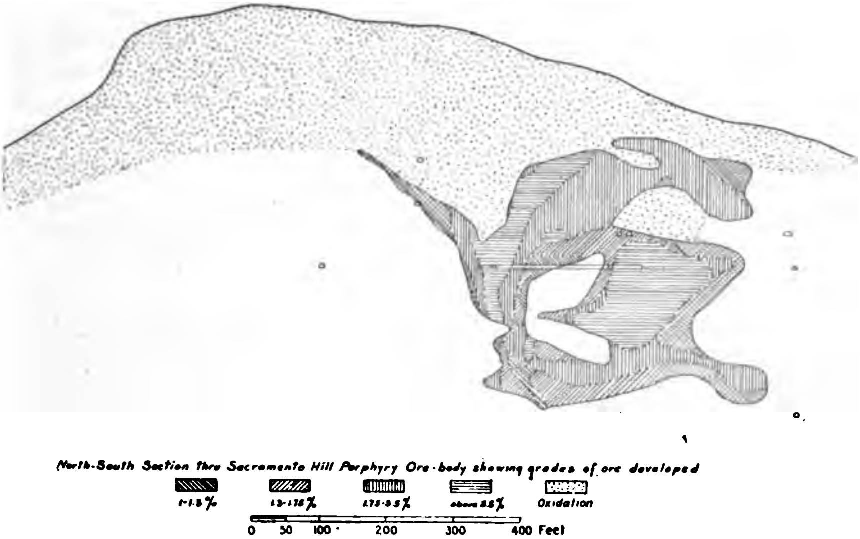

North-South Section thru Sacramento Hill Porphyry Ore-body showing grades of ore developed
1-1.5%
1.5-1.75%
1.75-3.5%
above 3.5%
Oxidation
0 50 100 200 300 400 Feet

Plate 11.—Sacramento Hill Orebody.

Four convenient subdivisions can be made, dependent on the formation replaced, as follows: ores in porphyry, in contact breccia, in contact metamorphic limestone, and in relatively unaltered limestone. In considering these ores, the primary mineralization will be taken up first and the effects of meteoric water circulation left for later discussion.

Ores in Porphyry

To this type belongs the ore developed recently in Sacramento Hill. The ore occurs as a very irregular mass of chalcocite, pyrite and some bornite in brecciated, altered porphyry at the inner side of the contact breccia. The ore also penetrates the contact breccia, as shown in Plates 5 and 11, and connects with some typical bodies in this latter rock. The porphyry, in general, is sericitized, and somewhat silicified close to the breccia, and in parts of the ore. The primary sulphide minerals are pyrite and bornite alone, this being the only type of ore in the camp in which chalcopyrite is not an important constituent. Partial analyses of this porphyry ore are given in Table 2.

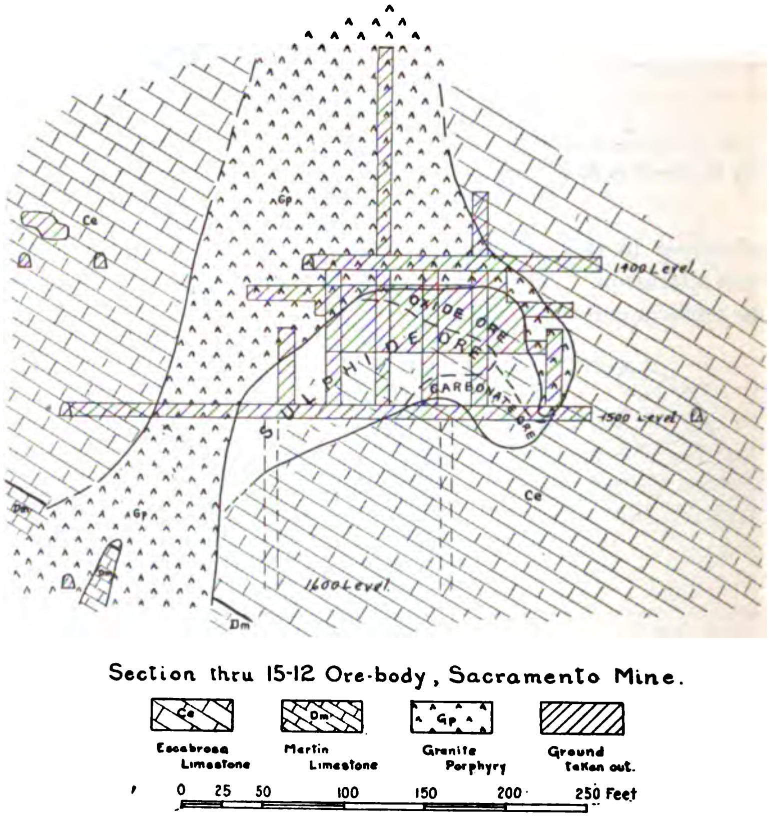


PLATE 12.—SHOWING CONTACT OREBODY ALONG SACRAMENTO DIKE AND SPECIAL FEATURES OF OXIDATION.

Besides the ore in Sacramento Hill, ore occurs in porphyry in a great many places in the contact-metamorphic zone, where the sulphides replace limestone and in a minor, but very distinct way, the adjacent porphyry. A noteworthy example of this is the orebody known as the Dividend Slice No. 1, where most of the porphyry is entirely converted into ore.

Ores in Contact Breccia

This second type of ore is found from the Holbrook to the Lowell, Oliver and Sacramento mines. The bodies are extremely irregular in shape and size, and are not confined to any definite horizon, but are usually found in unenriched and unoxidized state. One of these bodies outcropped to the west of Sacramento Hill.

The breccia, as has been explained, is highly silicified, and the ores themselves may be in an extra siliceous portion, or in a chloritized zone due to one of the later dikes mentioned before. The value of the contact breccia as an ore zone has been realized only within the last few years, as the rock is perhaps the most difficult of all to prospect.

Ores in Contact-Metamorphic Zone

To this class and the next belong a great majority of the ores heretofore mined in the district. The contact-metamorphic zone, marked by many alteration products in the limestone, but mainly by sericitization of both intrusive and intruded rocks, forms a block around Sacramento Hill and the associated extensions of the porphyry. The ores occur as irregular bodies intimately connected with dikes and sills which are direct offshoots of the main porphyry masses. The replacements extend from some distance in the porphyry, through some of the unrecognizable altered limestones, into the distinct bedding planes of less metamorphosed sediments.

Faulting and crushing previous to the introduction of the porphyry and ore solutions played an important part in determining the location and limiting the size of individual orebodies. Very little post-ore deposition faulting has occurred, so little in fact, that it is a negligible factor for most of the district.

The contact-metamorphic zone occurs almost entirely in Martin and in the first 200 ft. of Escabrosa limestone. Some orebodies, however, are found in the top 100 ft. of Abrigo as shown clearly in Plates 5, 6 and 7. To this zone belongs the ore found along the Dividend Fault, which has played such an important part in structure and mineralization of the district. The rich ore mined early from the Czar and Holbrook was taken out along this fault, and rich ore is still being mined in the Czar from along it.

To this type of orebody belong also a great part of the ores immediately adjacent to the Sacramento dike, Lowell sill, and to dikes in the

Junction and Oliver mines, though these bodies, being outside of the highly altered zone, generally grade into typical replacements of the unaltered limestones.

Ores in Relatively Unaltered Limestone

To this subdivision belong the orebodies in the Southwest, Higgins, Wolverine, Uncle Sam, Shattuck, the south end of the Spray, Irish Mag and Oliver mines, all of the ore in the Cole and Briggs, and the major portion of the ores in the Sacramento, Hoatson and Junction mines. A brief description of each of these occurrences is given below.

The Southwest orebodies, which include the original ore discovery of the camp, occur as replacements of Martin and Escabrosa limestones along the Czar Fault. The original Queen orebody outcropped at the surface, at the big glory hole at the base of Queen Hill. It occurred in the foot-wall side of the Czar Fault, replacing Martin and Escabrosa limestones. Following the Czar Fault to the southwest is a continuous chain of orebodies in the hanging-wall side of the fault, in the same stratigraphic horizon, until the present Southwest workings are reached. At this point the orebodies again cut across to the foot-wall side, but keep in the same horizon. It is these foot-wall orebodies that are being mined at present. The Czar Fault is here broken up and forms blocks with east-to-west faults. In this crushed zone occurs a large mass of brecciated iron-stained silica, which outcrops on the surface on both sides of Hendrick's Gulch. The orebodies are found at and near the contacts of this mass, as replacements of Martin and Escabrosa beds, and penetrate the breccia itself for short distances. The ore is both of copper and of lead.

The ore early mined along the Czar Fault (and now being largely remined), was closely associated with porphyry dikes and sills, which came up along the fault. The present orebodies, however, are not in contact with porphyry. The silica breccia mass, however, in depth is connected with porphyry.

Following the Silver Bear Fault, which forms the eastern boundary of the Queen Hill Block, another series of orebodies occur, also as replacements of Martin and Escabrosa limestone. These orebodies are directly related to porphyry dikes and sills.

The Higgins, Wolverine, Shattuck and Uncle Sam orebodies occur close to and in contact with the Shattuck dike. The Higgins and Wolverine are the farthest west, and therefore farthest removed from Sacramento Hill. Here the dike, as exposed underground, occurs as a dike of unaltered porphyry, and the orebodies are found on both sides replacing Abrigo limestone about 100 ft. below the top. The Uncle Sam and Shattuck orebodies occur as replacements of Martin and Escabrosa limestones, very intimately associated with a large sill of porphyry into which the Shattuck dike makes in depth. At the Shattuck, another mass of silica breccia occurs, more intimately associated with the porphyry than at the Southwest, with ore formed at the contacts, and penetrating into the mass. Here again the ore is both lead and copper as at the Southwest.

To the south and west of the Contact Block is a relatively up-thrown block, bounded by the Dixie-Howell-Senator Fault on the north and east, and by the Oliver Fault to the southwest, previously mentioned as the Oliver Block. In Irish Mag, Gardner, and Oliver ground, the block is much broken up by N.-S. fractures, and here there are orebodies in the foot-wall side of the Dixie-Howell-Senator Fault replacing Abrigo and Martin beds. In Spray, and in part of the Oliver ground where the block is less fractured, orebodies are found entirely as replacements of Abrigo limestone, from 100 to 300 ft. down from the top. In Spray ground no porphyry has yet been found anywhere near these orebodies. In Oliver ground, they are associated with small dikes.

In the Cole mine, orebodies occur in the Martin limestone and extend for a short distance into the top of the Abrigo, along N.-S. fractures. These orebodies occur in the hanging-wall side of the Oliver Fault. This fault, between the Cole and Oliver mines, has cut off one of these orebodies and is, therefore, post-ore deposition in age, one of the few breaks of its kind in the productive area of the camp. The Cole orebodies are not directly connected with porphyry.

The orebodies of the Briggs mine are similar to those of the Cole in structure, being related to a N.-S. fractured zone, not directly connected with porphyry. They are also replacements of Martin limestone.

The orebodies in the Hardscrabble claim of the Sacramento mine, and part of the Hoatson mine are directly associated with the Sacramento Dike. The orebodies exist as replacements of Escabrosa limestone, the horizon being about 200 ft. up from the base. The orebodies here are in direct contact with the dike on both sides, and make into it for short distances.

To summarize, the orebodies in the unaltered limestone occur for the most part at or near porphyry contacts, but they also occur away from porphyry in fractured country. In fractured country and where intrusions are numerous, the ore horizon tends to rise, and ore is found mostly occurring from the base of the Martin limestone to 300 ft. above the base of the Escabrosa. In unfractured, and relatively less intruded country, on the other hand, the horizon tends to drop and ore is found in the Abrigo limestone.

In the illustrations showing the zones of alteration (Plates 5*A*, 5*B*, 6*A*, 6*B*, 7*A*, and 7*B*) the orebodies in porphyry are seen to be between the sericitized and silicified portions of that formation. The ores in breccia are in a highly silicified zone. The ores in the contact-metamorphic limestones occur where the alteration is mainly sericitic.

Table 2 gives partial analyses of the orebodies along sections shown by the illustrations and the rocks that they are found in, as well as the character of the ore dependent on oxidation.

TABLE 2.—*Partial Analyses of Orebodies along Sections of Plate* 4

Section A-A

Rock Replaced	Location	Oxidation Features	Cu, Per Cent.	Fe, Per Cent.	S, Per Cent.	SiO_2, Per Cent.	Al_2O_3, Per Cent.	CaO, Per Cent.
Porphyry	East end Sac. Hill	Enriched	2.76	9.3	8.0	59.6	13.6	1.2
Porphyry	West end Sac. Hill	Enriched	1.8	9.4	9.8	60.5	11.4	0.9
Porphyry	West end Sac. Hill	Enriched	3.7	10.8	11.8	58.1	10.4	0.8
Porphyry	West end Sac. Hill	Enriched	6.6	13.6	15.8	50.1	7.8	0.7
Porphyry	West end Sac. Hill	Enriched	7.8	17.0	20.1	44.3	6.0	0.7
Porphyry	West end Sac. Hill	Enriched	12.2	16.9	21.3	43.7	3.8	0.4
Contact breccia	Breccia West of Sac. Hill.	Primary	6.4	19.1	22.4	40.7	5.3	0.7
Contact breccia	Breccia West of Sac. Hill.	Slightly enriched	6.1	14.7	16.4	46.1	9.1	0.5
Contact metamorphic limestone	Holbrook	Enriched	8.8	22.7	?	21.8	15.1	0.9
Contact metamorphic limestone	Holbrook	Enriched	4.4	24.4	25.3	13.0	15.1	1.0
Contact metamorphic limestone (slightly altered)	Holbrook	Slightly enriched	5.0	31.1	35.2	15.2	6.2	
Escabrosa limestone	Holbrook	Residual carbonate	6.3	22.3	0.2	10.1	5.3	14.8
Devonian limestone	Southwest	Residual carbonate	4.1	12.2	0.1	18.3	7.3	12.9
Section B-B								
Escabrosa lime and porphyry	Uncle Sam Mine	Residual ore	14.7	26.3	...	20.6	7.5	2.6
Devonian lime	Uncle Sam Mine	Residual ore	6.5	22.5	...	19.8	9.5	7.0
Abrigo limestone	Gardner	Primary	6.1	28.5	29.3	20.7	4.3	1.7
Devonian limestone	Gardner	Primary	4.7	32.5	36.0	16.1	6.1	3.1
Escabrosa limestone	Gardner	Residual carbonate	8.6	25.9	...	20.0	5.1	3.0
Metamorphosed Devonian	Gardner	Primary	4.3	34.1	27.7	14.8	2.45	1.3
Contact breccia	Gardner	Primary	4.8	30.6	40.0	13.6	4.5	1.4
Metamorphic limestone	Sacramento	Enriched ore	9.3	22.7	23.0	25.6	11.4	1.3
Section C-C								
Abrigo limestone	White Tail Deer	Residual carbonate	5.25	13.8	...	45.9	5.2	2.6
Partly altered Devonian	Holbrook	Residual carbonate	8.1	13.2	Tr.	26.0	10.5	1.6
Partly altered Escabrosa	Holbrook	Residual carbonate	4.4	25.1	Tr.	19.4	9.5	1.5
Sericitised metamorphic	Holbrook	Enriched sulphide	5.5	21.1	?	14.9	23.0	0.4
Sericitised metamorphic lime and porphyry	Holbrook (Dividend).	Enriched sulphide	6.6	17.5	12.6	17.8	23.2	0.3

MINERALOGY OF THE ORES

As in the case of rock alteration, primary minerals in the ores have been changed greatly by the circulation of meteoric waters, and the criteria have already been given for determining the primary zone under the caption, "The Effect of Contact and Hydrothermal Metamorphism on the Formations."

The original sulphides of copper deposited are comparatively few in number, chalcopyrite and bornite being the important ones economically. Tennantite is the only other copper-bearing mineral so far found in the primary zone, if exception be made of one or two unknown minerals seen with the microscope. Pyrite is such an important constituent of the ores, that it must be considered along with the copper minerals, and magnetite, hematite, sphalerite and galena are common enough to be considered part of the ores, especially the last two, since they are the source of oxidized ores of importance.

Chalcocite is very abundant in many forms in the zone of enrichment but has not been seen in the undoubted primary zone, though this has been considerably explored. It is therefore highly probable that this mineral was not formed by the ascending solutions in this district, and conflicting microscopic evidence in the ores above has been considered insufficient to offset the field evidence of the primary zone.

Pyrite

Of the sulphides, pyrite is generally the first formed, and accompanies every one of the stages of alteration described under metamorphism. In the ores, very fine examples of the structures described by Graton and Murdoch[8] may be seen, grading from the perfect "porphyritic" relation between well-formed pyrite crystals and the other sulphides, to a mass of pyrite grains with little interstitial chalcopyrite or bornite.

As additional evidence for explaining the mode of formation of some of the sulphides the following observations will be given. Around any orebody, pyrite usually persists in the same general relations to the rock or gangue, that it has to the other sulphides in the ore, with the exception that in the ore the individual pyrite grains are broken up, penetrated or reabsorbed by the other sulphides in such forms as are illustrated in Plate 13, Fig. 2, and Plate 14. Thus, ore in porphyry has pyrite veinlets extending into the rock along joints and it will be found disseminated through the rock in irregular grains that usually started around a biotite. These same pyrite veinlets and disseminated grains will persist even when all the rock has been replaced by sulphides, as can be seen in Plate 13, Fig. 3. In limestone replacements, pyrite will extend out along bedding planes in layers that are continuous with pyrite bands in the copper ore. And pyrite crystals will be found to have the same "porphyritic" relation to slightly altered Abrigo limestone as they have to the rich ore in

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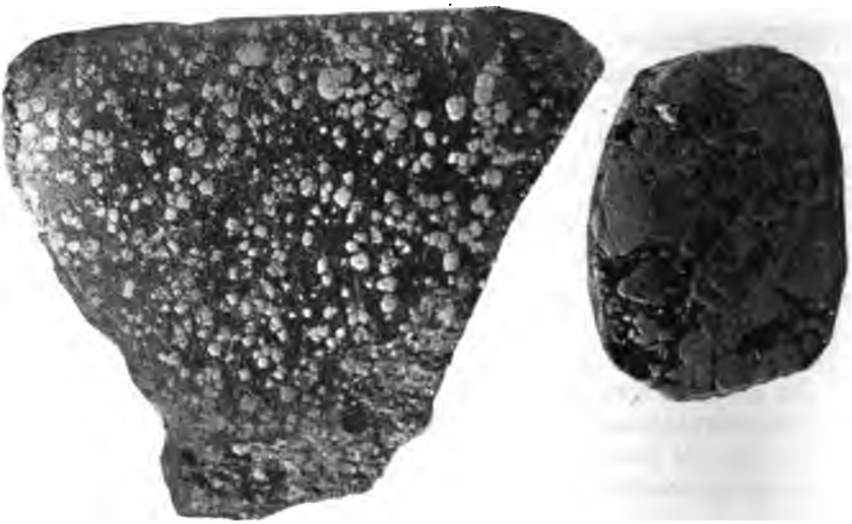
[8] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper, *Transactions*, vol. 45, pp. 26–93 (1913).



Figs. 1 and 2.—"Porphyritic" Structure of Ores in Abrigo Limestone (left) and in Breccia.

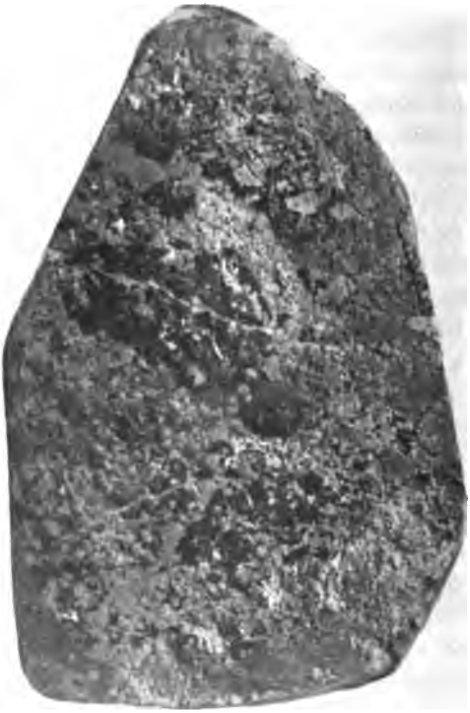


Fig. 3.—Replacement of All Rock by Sulphides with Pyrite Veinlets Persisting.

Plate 13.

that formation, with the exception given above, that there is little breaking up of the pyrite (see Plate 13, Fig. 1).

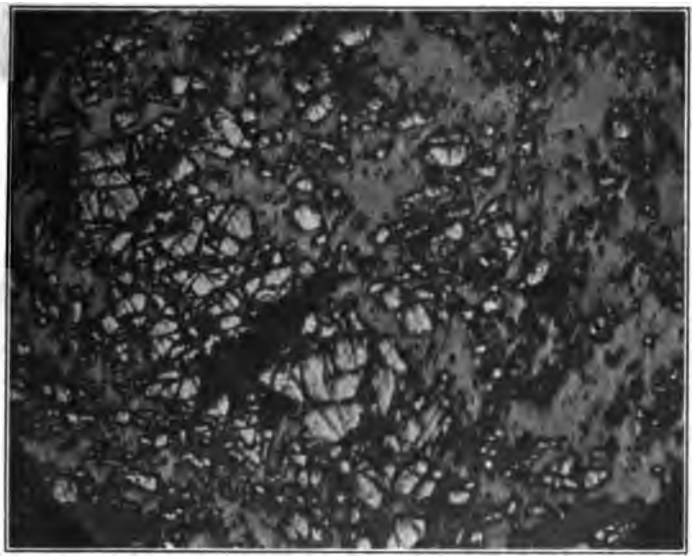


Fig. 1.

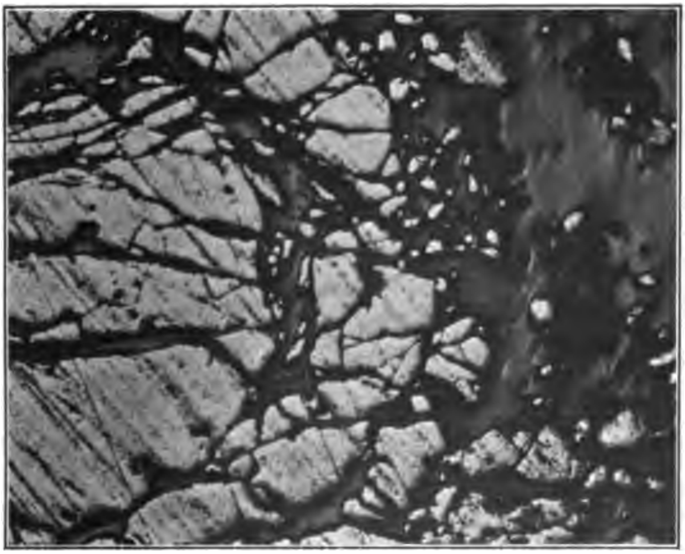


Fig. 2.

Plate 14.—Original Pyrite Grains Broken and Reabsorbed by Other Primary Sulphides.

This information, coupled with microscopic observations concerning the age of crystallization of the pyrite has led to the belief that in the ore formation pyrite is the advance sulphide, formed partly by solutions that are not yet precipitating copper minerals. When these latter are formed, they take their place by replacement of the gangue and rock mostly, but also by slightly replacing the earlier formed pyrite. This may explain certain veinlet-like stringers of pyrite, which apparently cut the other sulphides, but which when examined in detail show that each grain of pyrite has been attacked by chalcopyrite or bornite.

Age Relationship of Pyrite.—In the porphyry ores, pyrite is found to have crystallized at about the same time as the sericitization occurred with some of the grains a little earlier. Pyrite deposition also extended into the period of silicification, but was undoubtedly also replaced by quartz. Bornite, the copper mineral, is found replacing the pyrite as already mentioned.

In the contact breccia, due to the many complicated interrelations found, the pyrite cannot be said to have formed at any definite stage. In general it holds true that pyrite is earlier than the rest of the sulphides, but distinct veins of it have been found through crushed pyrite, bornite and chalcopyrite ore. Magnetite, specularite, and sphalerite are of about the same age as the pyrite, and garnet, diopside and wollastonite slightly earlier.

In the contact-metamorphic limestone orebodies, pyrite and sericite again belong to the same period of formation, earlier than the copper sulphides, and slightly later than tremolite and other contact silicates. In this zone, pyrite veins are found cutting magnetite, specularite and earlier pyrite bodies.

In the orebodies that replace unaltered limestone the relations are the simplest, pyrite being closer in age to the copper, zinc, and lead sulphides than in any other place.

Chalcopyrite

This mineral is undoubtedly the most abundant primary copper sulphide in the district, and is prevalent in all the ores, except in the disseminated porphyry ore in Sacramento Hill. This ore is distinguished mineralogically from all others in the district by having as undoubted primary sulphides, pyrite and bornite alone.

Chalcopyrite is generally associated with the more siliceous ores, except in porphyry, its period of formation being slightly later than that of sericitization, but contemporaneous with silicification. In cases where there is late chloritization, some remarkably pure bodies of chalcopyrite are formed, as is also the case in certain vein-like bodies extending far above the general ore horizon, where galena is also abundant.

The intergrowths of chalcopyrite and bornite show them to be of contemporaneous precipitation, with a slight lagging of the bornite formation.

Bornite

This mineral is universally found in the primary ores, though not as abundantly as chalcopyrite, as has been stated.

Although bornite occurs undoubtedly in the richer primary ores, there has been no reason found for its formation in preference to chalcopyrite, as it comes in rich and lean ores, and in those with a great amount of pyrite, also when there is scarcely any pyrite. Fig. 1, in Plate 18, illustrates a case where bornite and pyrite replace the cement in a calcareous Devonian sandstone, where the amount of copper is not enough to give 0.2 per cent.

Tennantite

This mineral is found usually in microscopic amounts only, in practically all the ores. Its age of crystallization is between that of pyrite and chalcopyrite. There may be some tetrahedrite, but as arsenic is more abundant than antimony in the specimens analysed, the mineral seen has been taken to be tennantite.

Sphalerite

The sulphide of zinc is found in all the primary ores in minor quantities, with zonal inclusions of chalcopyrite and galena in very fine grains (Plate 17, Fig. 1). In a few places such as between the Southwest and Czar mines, in the Gardner, and in Briggs ground, sphalerite is found in rather large bodies with pyrite and chalcopyrite. It generally replaces Martin limestone, and apparently its general relations are no different from those of the copper ores.

Galena

Like sphalerite, galena is associated with most of the primary ores, but is decidedly more abundant where the ores are in highly chloritized ground. The ground adjacent to the latest basic porphyry dikes, in contact breccia and in limestone, generally has more galena than the rest. An exception to this may have been the zones around the silica breccia masses of the Southwest and Shattuck mines, but unfortunately oxidation has obliterated thoroughly all primary mineralization. Lead carbonates and sulphates are here found in abundance.

Galena is found more abundantly also in the top zone of the ores, as a residual sulphide, so the general impression given by the occurrence of the lead sulphide is that it is more abundant in the last stages of mineralization and also that it was carried farther than the copper.

Other Primary Metallic Minerals

Magnetite and specularite occur sometimes in large bodies that show a very perfect intergrowth of these oxides, as well as with pyrite, as has been mentioned. The occurrence of these bodies is limited to the contact-metamorphic zone, or is immediately on the contact of porphyry dikes. A few grains of pyrrhotite have been seen here, indicating the high temperature of their formation.

Specularite is also found associated with the orebodies in contact breccia that come with the basic, chloritizing porphyry dikes. The oxide is here a valuable indication of the proximity of ore, in a zone where all other features are baffling in their irregularity.

Silver and gold minerals have not been seen in the primary zone, with the possible exception of a few grains of polybasite found in galena. Gold and silver values are considerable in all the copper ores, and more so probably in the sulphides than in the oxides, but their amount could easily escape detection even under the microscope. Tennantite, as well as one or two brownish unknown minerals in the sulphides may be the precious-metal carriers.

IX. OXIDATION AND ENRICHMENT

General Features

The most striking feature of oxidation of the orebodies in the Warren district is the varying elevations at which the column, from oxidized to primary ores, repeats itself in going from the western to the eastern side of the mining area. At the extreme western end, the Higgins and Wolverine mines have primary ores in Abrigo limestone far above the level of the adjacent creek bottoms, and consequently far above the present ground-water level. In the Czar mine oxidation and enrichment go down to just under the 300 level, which is about 5,000 ft. above sea level. In the Holbrook division some primary ore outcropped at the west of Sacramento Hill, while some distance south, enrichment goes down below the 500 level. In the Gardner, primary ores are not found till the 9th level is reached, and in the the Lowell mine they are encountered, in some instances at the 10th level, but generally below the 13th, although the permanent water level was encountered here around the 10th level, or at an elevation of 4,325 ft.

In the Hoatson mine, oxidation features were found down to an elevation of about 3,800 ft. above sea level, and even deeper in part of the Sacramento and Junction mines, although the permanent water level was far above this before the mines were drained by opening up and pumping from the deeper levels of the Junction.

The oxidation features shown by the illustrations of the sections through the camp (Plates 5*b*, 6*b*, and 7*b*), illustrate this general inclination to the southeast. It will be noticed that the oxidation follows in general the dip of the Paleozoic sediments, the average horizon reached being somewhere in the Devonian. There is a marked lagging of the oxidation at the eastern end, however, as here it does not reach the top Devonian. In this, the oxidation may be seen to correspond more closely to the pre-Cretaceous erosion surface.

It will be seen also that there are some sudden changes in the level of oxidation, as best shown in section A-A of Plate 5, and also that there are great variations in the extent of the zone of secondary sulphides. All these features have fairly definite relations to the geologic history and structure, to the nature of the formations, and to the primary alterations in them. The object of this chapter will be to explain what is known of those relations.

It will first be necessary to present the criteria for the subdivisions of the zone in which meteoric waters have circulated, in many directions, but generally downward. It has been stated before that the primary zone was marked by steady conditions and constant mineralogical associations. The deepest effects of the surface-water circulation are immediately and clearly marked in this district by veinlets, filled cavities and coatings of other minerals.

Just above the primary zone there is an extremely variable thickness where the prevailing secondary mineral is siderite with a little gypsum and an extremely small amount of chalcocite. Above this is usually a zone where halloysite and kaolin are most prominent among the secondary gangue minerals, and chalcocite is abundant as a replacement of primary sulphides. Next higher, limonite, and the staining due to iron oxides—gossan formation, in other words,—is the prevailing feature. Within this zone there are found the local bodies of residual ores, which have for some reason resisted the general leaching effect of strongly oxidizing waters. One or all the zones above the primary may be lacking at any point for reasons which will be explained later.

RELATIONS OF OXIDATION TO STRUCTURE AND GEOLOGICAL HISTORY

Notman[9] has already pointed out the relations between the attitude of the oxidation and present and past water levels. That the present water level does not account for the oxidation is proved by the occurrence of unoxidized ores above that level in formations that are in other places completely oxidized also above that water level. This is illustrated by the primary ores in Abrigo at the Higgins mine, and the completely oxidized Abrigo ores in the Wolverine and White Tail Deer mines.

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[9] *Transactions of the Institution of Mining and Metallurgy*, vol. 22, p. 561 (1913)

Further proof is had in the presence of gossans several hundred feet below the water level in the Lowell and Hoatson mines.

The general plane of the oxidation corresponds very closely to the known pre-Cretaceous erosion surface, and the tilt of the oxidation is, therefore, the present tilt of the Cretaceous sediments. If the ground-water level at the time of oxidation had been an inclined plane that followed the Devonian horizon, due to the permeability of the purer limestone above, there would not be abundant unoxidized orebodies in Escabrosa limestone in the Sacramento and Junction mines. And if the ground-water level had stood several hundred feet lower in those mines at some post-Cretaceous time and had risen, say, due to valley fillings, it is probable that the ground water, now directly over that rise, would stand horizontal, and would keep to the old plane, or even a lower one farther west, which is far from being the case.

There is field evidence of another form to show that the oxidation was pre-Cretaceous, and this is the presence of gossanized fragments of porphyry, as well as fresher ones at the base of the Glance Conglomerate. The exposed pre-Glance topography also shows hard gossan reefs around the southeast side of Sacramento Hill.

Additional weight is given to the idea of pre-Cretaceous oxidation by the geologic history of this region. A very long erosion and a consequently long oxidation period is recorded between Pennsylvanian and Cretaceous times by the disappearance of the paleozoic column off of the north side of the Dividend Fault. And while the granite porphyry intrusion may not have taken place till a good part was gone, the nature of the rock as well as of the later minerals formed presuppose a considerable covering of sediments which were removed by erosion before Cretaceous time.

It is, therefore, probable that the main part of oxidation and enrichment as found at present took place when the Paleozoic sediments were almost horizontal, the only tilt noticeable being slightly to the southeast.

There is very little evidence of superimposed oxidation since Cretaceous times, due very probably to the later ground-water level being higher than the first. Narrow, limonitic seams, and very slight enrichment on primary sulphides in the west end of the camp are the only visible effects of this oxidation. The post-ore faulting which is also post-oxidation in most cases, is not found very frequently in the thoroughly explored portion of the district, but will very probably be found more frequently when operations extend into other blocks.

RELATIONS OF OXIDATION TO THE FORMATIONS AND TO METAMORPHISM

The circulation of meteoric waters has been very much influenced by the material they traversed, and this was dependent upon the formation and the degree of alteration.

In the Carboniferous limestones the massive purer beds had a tendency to crack and break rather than to fold, and waters could dissolve out channels through which extensive oxidation could be carried down rapidly. The same applies to the purer lime beds in the Devonian and Cambrian formations.

The more impure beds, which are also more thinly laminated, had a tendency to crush and fold, and while as a whole they may be found more altered by meteoric waters, it is because they retarded and distributed the effects all through the rock. This is noticeable in the Martin and Abrigo formations especially, the Abrigo being by far the most impervious.

The ability of meteoric waters to circulate and oxidize quickly in the Escabrosa limestone has made possible the abundance of residual ores in this formation. These are mostly carbonates fixed in this form against total leaching by the continually circulating cold waters. In a great many cases, however, the residual ore is found in the form of sulphide, mostly chalcocite, and also, in rare instances, as the original chalcopyrite and bornite. These sulphides are called residual because they are far above the level of normal enriched sulphides, and they have limonitic gossans or carbonate and oxide ores under them. Usually they are very pure, rich sulphides of copper, and are in the primary zone of sulphide replacement where the adjacent rock is pure calcite and contains little or no pyrite. Occurrences like this are found in the limestones all the way to surface where the main ore zone may be a thousand feet below.

The same quick circulation through the Carboniferous limestones has usually brought the zone of complete oxidation directly against the primary ores, with a thin intervening zone of secondary sulphides, which, as will be shown later, is not one of enrichment, but of leaching of values. The slower circulation in the impurer Devonian and Cambrian produced fewer residual orebodies, but a deeper zone of enrichment.

In the porphyry and contact breccia the differences in the effect of secondary alteration are far more noticeable, as can be seen from Plates 5*b* and 6*b*. Both the porphyry and breccia retard the meteoric water circulation, the breccia almost entirely, and the porphyry making it slow enough so that sulphide enrichment is at its most efficient point.

A special case which illustrates the difference in the penetration of oxidation through different rocks is given in Plate 12, which is an east-to-west section through an orebody on the 1,500 level of the Sacramento mine, at present being extracted. This ore occurs along the Sacramento dike, here cutting Escabrosa limestone and having a projection downward to the east as shown. The ore replaces the limestone, which is not appreciably metamorphosed at a distance from the porphyry. The replacement is very perfect along the beds, as certain cherty bands are left untouched in the midst of the ore.

The oxidation feature that is noticeable is that the ore is almost unaltered close to the main dike, enriched farther away, converted into mixed oxide and sulphide under the thin portion of the porphyry, and entirely into carbonate next to the limestone. The water circulation here must have come partly from the east and partly from underneath through the limestone, as the porphyry ordinarily presented an effective barrier to circulation perpendicular to the beds before the tilting.

The oxidation features are even more noticeably related to the processes of primary ore alteration than they are to that of the formations.

Intense silicification such as is found in the contact breccia and in a great part of the contact metamorphic limestone of the Gardner mine made circulation so slow that in these places the primary ore zone remains at a level far above those of adjacent occurrences. Also, the oxidation is so complete where there is any meteoric circulation at all that the zone of leaching comes to within a few feet of the primary zone, with only a thin layer between where covellite is most abundantly found in the district. These relations are well shown in Plate 5, Section A–A just west of Sacramento Hill.

Sericitization produced rocks in which the circulation was slow, but in which the secondary waters had opportunity to remain acid, as nearly all the lime was removed in the primary alteration and pyrite was abundant all through the rocks, Under these conditions the depth of the zone of secondary enriched sulphides was greatest, and the enrichment most efficient. At the same time, in the rocks very much sericitized there are no residual carbonates found, due of course to the lack of lime. If, however, the ore formation extended into unaltered limestones above those sericitized, carbonates might be found higher, as is the case in the Uncle Sam mine. Native copper and oxides are developed to a great extent in a shallow zone between the gossans and enriched sulphides in aluminous ground due to primary sericitization.

The ores in Sacramento Hill porphyry, and in the zone of contact-metamorphic rocks around the main intrusives are the best examples of the coincidence of sericitization and deep sulphide enrichment, as can be seen in Plates *5a*, *5b*, *6a* and *6b*. The amount of enrichment even in these cases has not been exceedingly great, as the original ores were rich to begin with, and very pyritic bodies have remained poor, first because pyrite is replaced with so much more difficulty by chalcocite than bornite and chalcopyrite, and second, because pyritic masses are generally quite solid, more siliceous and impervious.

The ore developed in Sacramento Hill is extremely irregular in shape and grade and owes its irregularity more to primary mineralization than to secondary enrichment, because so far as has been seen, the relative amount of primary bornite left in parts at the top of the ore close to the oxidation line is about the same as at the deepest levels. A section showing the ore grades is given in Plate 11, worked out from all information available. This body has not yet been mined extensively. The grades of ore can be seen to bear very little relation to the limonitic oxidation.

The orebodies classed as sulphide replacements of unaltered limestones are ordinarily found in their primary state or as residual carbonates. Oxidation is rapid through unaltered rocks, and waters usually carry enough lime to fix the copper as carbonate. In such a zone, carbonates are apt to be found to the very top of the mineralized zone, many times above hundreds of feet of gossanized ground.

TABLE 3.—*Showing Efficiency of Enrichment by Descending Waters*

Orebody	State of Oxidation	Tons per Cu. Ft.	Pounds Cu per Ton	Pounds Cu per Cu. Ft.
	Sericitized Contact Metamorphic Zone			
14-7	Primary	0.1143	126.2	14.42
13-13	Enriched sulphide	0.1008	189.8	19.10
	Sulphide Replacement Orebodies in Abrigo			
12-3-18	Primary	0.117	104.0	12.17
13-10	Enriched	0.0984	147.0	14.46
White-Tail Deer	Residual carbonate	0.0578	142.0	8.20
	Sulphide Replacement Orebodies in Martin and Escabrosa			
9-8-58	Primary	0.1255	130.0	16.31
8-17-11	Enriched and oxide	0.098	112.0	10.97
8-17-4	Residual	0.0609	113.0	6.88
8-16-5, 8-16-16	Residual	0.0609	137.0	8.34
8-16-18	Residual	0.0609	108.0	6.58
	Sulphide Replacement and Contact Bodies in Escabrosa			
1300 Junction	Primary	0.1111	116.0	12.90
16-3-39	Primary	0.140	140.2	19.63
16-3-68	Partly enriched	0.1267	192.4	24.38
15-12 Sulphide	Secondary sulphide	0.115	130.0	14.90
15-12 Oxide	Oxide	0.08	224.0	17.92
16-3 Oxide	Oxide	0.08	268.0	21.40
15-6-1, 15-6-9	Residual carbonates	0.065	170.0	11.05
15-13	Residual carbonates	0.065	188.0	12.20
14-20	Residual carbonates	0.065	114.0	7.41

Table 3 has been prepared to show approximately the efficiency (or lack of it) of enrichment due to descending meteoric waters.

Orebodies have been taken for analysis which were originally of similar composition, as far as knowledge of them goes, and from mining records kept for a period of several years, the contents of copper in pounds per cubic foot in place has been figured. This is of course a figure for the ore mined, and must differ somewhat from the actual content of all the ground in an orebody, depending on the method of extraction used. All errors have been considered, however, and the relative value of the figures is about correct. Unfortunately the ores in the breccia and silicified ground are all primary and no comparisons can be given, and all the ore in porphyry is enriched.

From these tables it will be seen that the copper content per unit of volume is increased materially in the sericitized contact zone and in the more aluminous Abrigo formation where the secondary sulphides are found. Chalcocite formation is accompanied by leaching out of values in the replacement bodies, and all the residual ores show dissipation of copper contents. The narrow oxide and native copper zone is generally the richest of all.

GOSSANS

In regard to the gossans formed in the different zones of primary alteration, they can be said to extend downward for the greatest distances in the more permeable ore zones—this is true for those classed as sulphide replacements. These gossans are likely to contain residual ores anywhere. The thoroughly oxidized portions, where the formations were silicified or sericitized or chloritized are apt to be more completely leached out, and penetrate the ore zone least in highly quartzose ground. In composition the gossans of course reflect their source.

The only generalization that can be made about gossans derived from lean sulphides and from pyritic copper ores is as follows. The mixed sulphides of copper and iron are more readily attacked by oxidizing solutions than the straight iron sulphides, therefore producing stronger solutions. These stronger solutions have been found to have a segregating action on the components of the surrounding rocks, separating to some extent silica, alumina, and lime, and also depositing the iron oxides in purer form. The gossans from pyrite alone have been observed to leave the gangues and oxides of iron in about the same mixtures as in the original primary material.

These observations are of course susceptible of an infinite number of variations depending on the original composition of sulphides, of their gangues, and of the surrounding rocks, but in general they can be used to some extent in judging the kind of ground under an oxidized portion of an ore zone.

MINERALS OF THE ZONES OF ENRICHMENT AND OXIDATION

Siderite

Under the zone of secondary enrichment the presence of siderite has already been noted. This mineral occurs sometimes in veinlets replacing either sulphides or chloritic minerals or calcite. At other times siderite is found in a curious box-work honeycomb structure as if it had replaced laminated rock along certain planes and the intervening material had been later dissolved out. The ferrous carbonate has a distinct selective action in replacing a rock, as it usually leaves intact pyrite grains, which then appear contemporaneous with the siderite. Sphalerite and galena are also found more abundantly with the siderite than around in the unaltered formation, and this fact has pointed to the possibility that these two sulphides may have been here deposited by secondary solutions.

Copper minerals in the siderite zone are very faintly coated with chalcocite. Gypsum is also found in vugs, even deeper than siderite, but apparently in late circulation channels.

Siderite probably formed as an advance mineral, as oxidation progressed downward. This may be proven by the closely similar forms taken by limonite in some gossans above enrichment, and the presence of siderite cores in that limonite. The iron carbonate has played an important economic rôle in serving as a precipitant for rich cuprite ores in the Czar mine, where copper-laden solutions have encountered a flat bed in Devonian limestone previously replaced by siderite and kaolin, and have precipitated cuprite and native copper in considerable amount. Throughout the district cuprite crystals can be found on oxidized or partially oxidized siderite.

Chalcocite

In the zone of sulphide enrichment chalcocite was the only secondary mineral of copper formed to any extent. It was usually formed by replacement of the bornite and chalcopyrite alone, leaving the pyrite untouched, or only slightly coated. The broken up, replaced pyrite grains in the primary ore remain in just about the same amount and with the same relations to chalcocite that they had to the primary minerals. When chalcocite and pyrite are the only remaining sulphides, it is difficult to believe that the iron mineral has not been abundantly attacked by secondary action. Pyrite has been replaced to a considerable extent, however, in the enriched ores of the contact-metamorphic type, by a sooty variety of chalcocite that possibly contains considerable covellite, since the latter has been seen coating pyrite grains in these ores.

In replacing the primary minerals chalcocite behaved in different ways depending on the general feature of the orebody. In the ore of

Sacramento Hill, pyrite is barely coated in the very sericitic portions of the porphyry, and is not affected appreciably in silicified rock. The selective action in replacing bornite is better marked here than anywhere else as is also the replacement along crystallographic lines of the bornite in the arrangement commonly known as "lattice structure." This form of alteration grades into the usual one of veinlets and irregular remnants, as is shown in Plate 15. The field occurrence of this "lattice

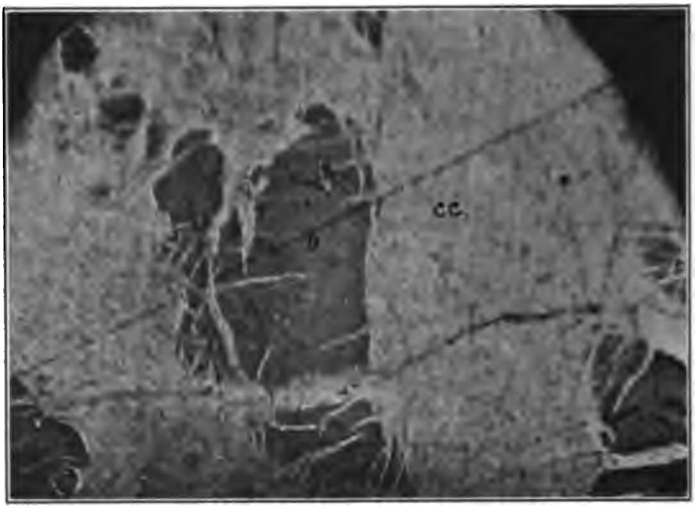

Fig. 1.

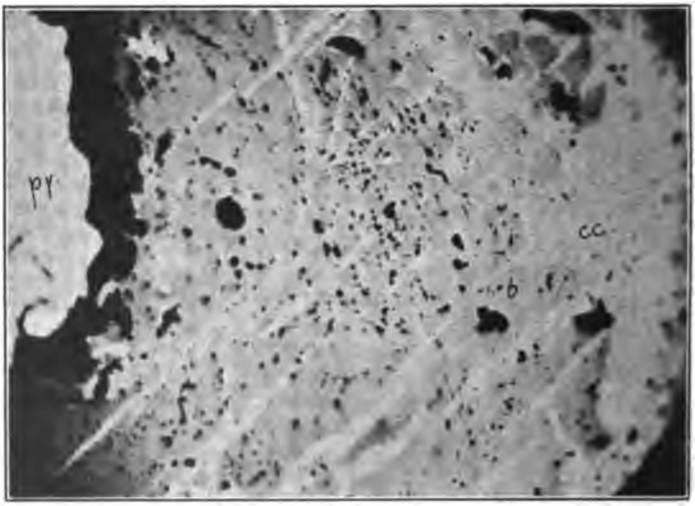

Fig. 2.

Plate 15.—Chalcocite, *cc*, Replacing Bornite, *b*. Pyrite, *py*.

structure" alteration has been quite carefully followed in the district and found everywhere to coincide with slow or incipient enrichment in rich ores, and where the selective enrichment of bornite in place of chalcopyrite is well marked. This has led to the conclusion that it is a delicate process in which crystallographic weakness can be of some influence. Etching of the chalcocite thus formed shows isometric figures, probably due to included or dissolved cupric sulphide,[10] but there are also stringers in which the chalcocite has orthorhombic cleavage, and these stringers can be seen to form a network of veins. The proximity and association of this orthorhombic chalcocite to major cracks and circulation channels, where even limonite is formed, points to its being due probably to the working over and recrystallization of the copper sulphides first derived from bornite.

Chalcocite of the "sooty" variety is found most commonly in the enriched ores of the zone of contact-metamorphic limestones. Massive chalcocite is here rare, and then only at the top of the zone of enrichment, with considerable native copper present.

In the ores that replace limestone, the formation of chalcocite is accompanied by leaching and formation of abundant cavities in the ore, a process that is reflected in the decrease of values as previously shown. This same thing is true in ores that have a very siliceous gangue, such as those in contact breccia. Pyrite is here oxidized to limonite even before the copper minerals are replaced by chalcocite, as is shown in Plate 16.

In the Southwest mine there is an occurrence of chalcocite that is somewhat different from the usual. The Devonian-Escabrosa contact beds are replaced by silica, almost chalcedonic in character, accompanied by abundant fine flaky specularite. This siliceous mass connects up with the main silica breccia replacement in this mine.

The top Devonian beds under those silicified have been hardly altered at all, except for a very slight amount of quartz addition, some recrystallization of calcite in stringers, and the formation of very fine veinlets as well as disseminated grains about a millimeter in diameter of what was apparently bornite and chalcopyrite, now replaced almost entirely by chalcocite. The original copper minerals have been found partially replaced by chalcocite in the center of coarse calcite veins that protected them. Pyrite is very rarely found.

The formation of chalcocite has taken place in this instance under conditions which would hardly allow the circulation of acid waters. A considerable amount of carbonates has formed, but in general the sulphide is relatively abundant. The possibility of chalcocite formation with the acid generated in the replaced mineral itself is indicated in this orebody, though this is by no means perfectly clear.

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[10] Posnjak, Allen and Merwin: *Economic Geology*, vol. 10, No. 6.

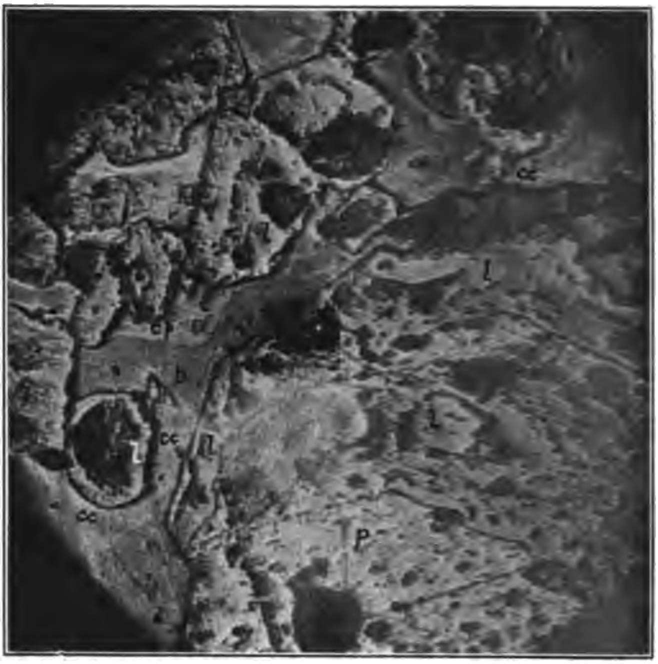

FIG. 1.

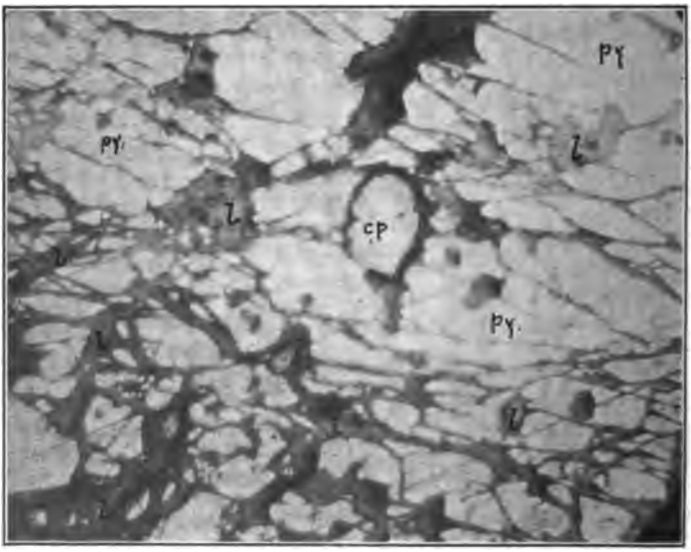

FIG. 2.

PLATE 16.—SHOWING OXIDATION OF PYRITE, *py*, TO LIMONITE, *l*, BEFORE REPLACEMENT OF COPPER MINERALS BY CHALCOCITE, *cc*.

Covellite

Covellite is formed in very minor amounts especially in the narrow oxidized zone above the contact breccia ores. It replaces chalcopyrite in preference to bornite, and is found always in microscopic needles. The only other occurrences of this mineral are where the enriched zone touches highly silicified, not very sericitic ores.

Bornite and chalcopyrite have been found in many ores of the district, formed by secondary agencies, as transition products, but they are of no economic importance.

Gangue Minerals

The gangue minerals that accompany the enriched sulphides are halloysite, kaolin, gibbsite, alunite, and serpentine with mixtures of quartz, calcite and the oxides of iron. Of the aluminous minerals, halloysite seems to be the most common, especially in the zone of highly altered limestones. Where the rock is sericitized so that the mica flakes are well formed, sericite persists unaltered through the process of enrichment and oxidation.

Oxide Ores and Native Copper

In the oxide ores just above the secondary sulphides, cuprite, native copper, tennorite, melanochalcite and delafossite are found, but the first two are the only ones of economic importance. The association of cuprite and siderite has already been given. The zone of oxide has in many instances encroached entirely on the sulphides, leaving only a remnant here and there of the latter. Flat orebodies along the limestone bedding are especially susceptible to this process.

The minerals of the residual orebodies are mostly the carbonates malachite and azurite, with some chrysocolla, aurichalcite and brochantite. Besides these, there are the sulphides already mentioned.

Carbonates

The association and the occurrence of the carbonates has been very well described by Dr. Douglas[11] and by Ransome[12] and no additional features can be given, except to repeat that these minerals are found all through the oxidized portion of the ores that replaced unaltered sediments, and are still a very important part of the ore reserves of the district.

Other Ores

In recent years oxidized lead ores have become more and more important, especially from around highly siliceous replacements in Martin and

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[11] *Transactions*, vol. 29, pp. 511–546 (1900).

[12] *Loc. cit.*, p. 125.

Escabrosa limestone. Most of this ore has come from the Southwest mine and occurs at the edge of the siliceous breccia, as cerussite with some anglesite and variable silver and gold values. The primary minerals from which this ore was derived are unfortunately not present now, but

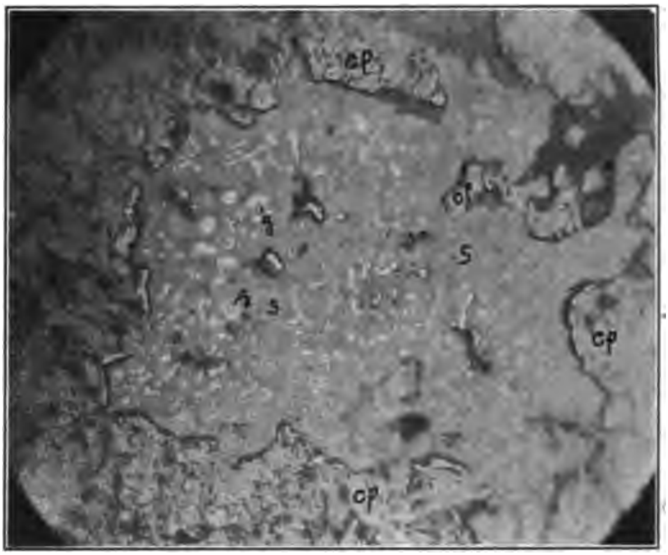

Fig. 1.—Sphalerite, *s*, and Galena, *g*, Included in Grains of Chalcopyrite, *cp*.

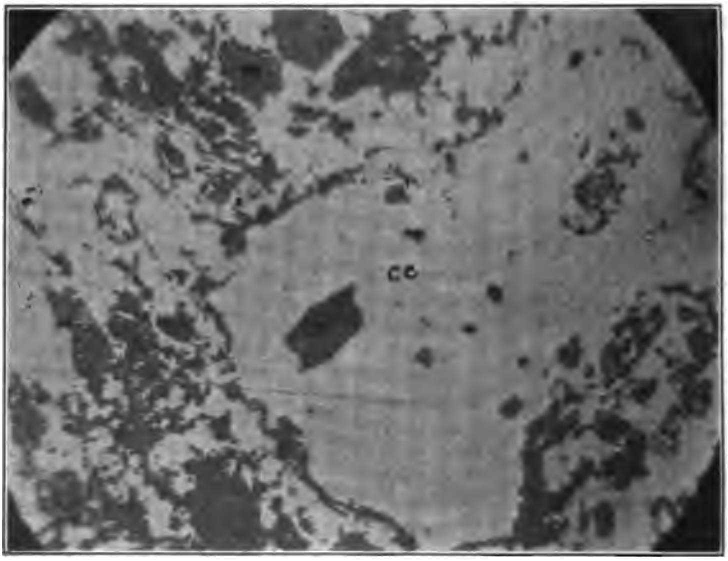

Fig. 2.—Chalcocite and Malachite. Alteration Resembling Intergrowth.

Plate 17.

it is very probable that the main one was galena, which, as has been before stated, is apt to come in vein-like masses above the general level of the copper ores. Lead ores have also been found with similar associations in the Gardner, Lowell and Briggs mines.

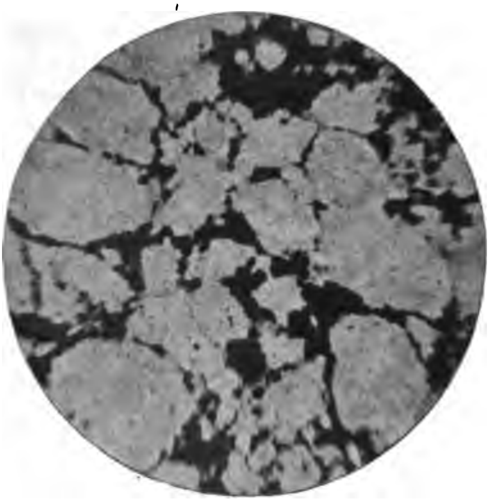

FIG. 1.—BORNITE AND PYRITE REPLACING THE CEMENT IN A CALCAREOUS SANDSTONE.

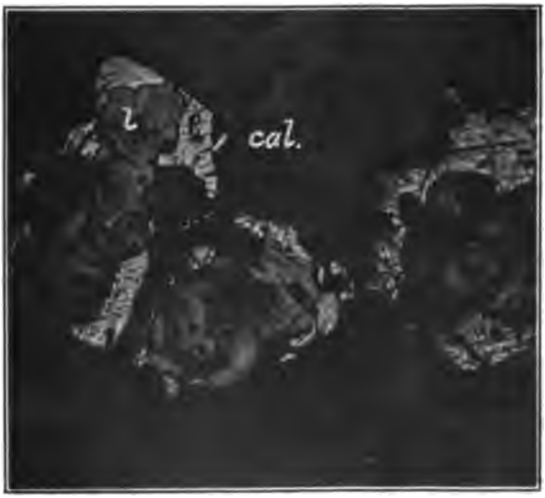

FIG. 2.—PYRITE REPLACED BY LIMONITE.
PLATE 18.

Vanadium, in the form of cuprodescloizite has been found in the Shattuck mine in considerable amount, and rarely in the Dallas and Sacramento. The occurrence may prove in future to be of economic importance.

Manganese ore, in the form of psilomelane and braunite, with little pyrolusite, has been mined lately from several places on the surface of the district. It is found in irregular bunches as a replacement of limestone beds in the Naco or Escabrosa, close to the contact breccia, or to highly silicified outcrops of the sediments. The origin of this manganese is not yet known, for although manganese is found quite abundantly with some of the carbonate ores underground, it is here in the form of soft earthy pyrolussite due apparently to the concentrating action of oxidizing water.

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ARTHUR NOTMAN: The Copper Queen Mines and Works, Part II. *Transactions of the Institution of Mining & Metallurgy*, vol. 22, pp. 550–562 (1913).
W. L. TOVOTE: Bisbee, a Geological Sketch. *Mining & Scientific Press*, vol. 102, pp. 203–208 (February, 1911).

Mention must also be made of the report on the mining geology of the Copper Queen property by John Mason Boutwell prepared in 1908–1909 for the use of the mines, one of the results of this work being the establishment of the present Geological Department.

ACKNOWLEDGMENTS

To the Copper Queen Consolidated Mining Co. for its support and encouragement in the preparation of this report, the greatest thanks are due.

Arthur Notman, Chief Geologist of the Company, has lent the utmost assistance, contributing from his store of knowledge gained by long and careful investigation in the district and in this part of the Southwest as a whole.

Former members of the Geological Department, and our colleagues of the Calumet & Arizona Mining Co., have also contributed materially to the information presented.

To Harvard University, and Prof. L. C. Graton in particular, we are indebted for being able to present the result of extensive petrographic and metallographic investigation, as 2,000 specimens from the Copper Queen geological collection were studied by one of the authors in the geological laboratories at Cambridge. Some of the photomicrographs were taken at the University of Arizona, for which thanks are especially due to Prof. C. H. Clapp.

Assistance has been received from the engineering department of the company in the way of maps and stope records, and from the chemical department in the form of the analyses that are given in the report. Thanks are due to the operating department of the mines for a great deal of information, also for coöperation in pointing out and solving most of the geological problems of the district.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Mine and Mill Plant of the Inspiration Consolidated Copper Co.

BY H. KENYON BURCH, B. S.,* MIAMI ARIZ.

(Arizona Meeting, September, 1916)

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* Chief Engineer, Inspiration Consolidated Copper Co.

INTRODUCTION

THE Inspiration Consolidated Copper Co.'s plant at Miami, Ariz., was designed and built to make possible the profitable working of a low-grade, finely disseminated copper deposit containing 100,000,000 tons of ore averaging 1.64 per cent. in copper.

From the beginning it was evident that the plant could not be kept integral but that a break would have to be made somewhere in the flow-sheet, removing at least the concentrator to a site more suitable than any available near the mine. It was finally decided, after considering numerous arrangements, to do the coarse crushing at the mine, to store the crushed ore in a bin from which it could be loaded into railroad cars and to haul it to the concentrator, an excellent site for which was available about 1¾ miles from the mine.

The original intention was to equip a plant to treat 7,500 tons of ore per day, but through the acquisition and proving up of additional ore reserves, the introduction of the Ohio caving system, and the excellent results obtained in the test mill (which made it possible to treat a lower-grade ore than had been thought possible) it was evident that a plant of much greater capacity should be supplied. It was, therefore, decided to treat approximately 15,000 tons of ore per day; the duties of the four main divisions to be as follows:

Division	Operating Time in Hours	Capacity in Tons per Hour	Daily Capacity in Tons	Available Capacity in Tons
Hoisting plant	15	1,000	15,000	25,000
Crushing plant	15	1,000	15,000	
Storage bin	....	......	......	
Concentrator	24	625	15,000	

It was realized from the beginning that with such an enormous tonnage to treat it would be well worth the time and cost to carefully work out a flow-sheet. Accordingly, a gravity test mill was erected and placed in operation near the Joe Bush shaft in November, 1910, and its operation was continued until August, 1911. Soon after this, flotation began to attract considerable attention in this country and realizing the possibilities that might arise through its systematic investigation, it was deemed advisable to go into the process in detail. A 50-ton Minerals Separation machine was first erected, which after a few months thoroughly demonstrated that flotation was applicable to the concentration of the Inspiration ores, but in order to better determine its proper place in the flow-sheet, a 600-ton test mill was designed and erected in 1913 on the benches of the concentrator site, the grading for which was at this time completed. Numerous flow-sheets were experimented with, the final result being the one now in use in the concentrator. A discussion of certain phases of the results accomplished in the test mill will be given in another section, mention being made of it here simply for preserving the proper sequence.

That the large-scale test-mill method for working out flow-sheets for large plants is the only logical method, is evidenced by the fact that in nearly every stage of the treatment either an entirely new machine has been adopted or a new application has been made of a standard machine; the result in each case being either increased efficiency or a more economical arrangement. The first mill was to have a capacity of 7,500 tons per 24 hr. This design covered an area of approximately 350,000 sq. ft., or a little over 8 acres. The mill now in use covers an area of approximately 125,000 sq. ft., or a little less than 3 acres, but has a capacity of 15,000 tons. The recovery in the two types of plants on favorable ores, that is, ores not carrying over 10 points of oxide, may be closely estimated at 70 per cent. for the first, and 85 per cent. for the second.

In order to account for the long period required for the design and construction of the plant, it may be interesting to note that six complete designs for the concentrator were executed, the idea being to keep this work abreast of the developments brought out by the test mill. Design No. 2 was completed and a contract entered into for structural-steel requirements. A portion of the steel had been fabricated when flotation developments pointed to the fact that wet gravity concentration could be greatly improved upon. At this point all work on the steel contract was stopped and that portion of the contract pertaining to the concentrator was cancelled. Although facts relative to flotation continued to develop, in July, 1913, the steel design for the present concentrator building was completed. This building was no sooner erected than very marked changes in grinding machinery began to develop, the ultimate result of which was another altogether new arrangement for the entire mill. As it stands today there are but three single pieces of machinery in the mill building occupying the places originally intended for them, these being the three electric cranes now in use. Considering these changes it is remarkable that such a good arrangement was found possible.

PLANT SITES

The sites for the plants as finally decided upon permitted of an arrangement entirely adequate to meet the proposed requirements, but the following important features may be noted:

The mine plant site is contiguous to the orebodies but at a safe distance from ground to be caved; it occupies a position as regards elevation well suited to a favorable hoisting arrangement and allows of a downgrade to the concentrator of 0.3 per cent. The shortest line from the shafts to the orebodies approximately bisects them, making possible a direct underground haulage system; the site occupies a remote corner of the property and does not lie in the general trend of the mineralized zone. The mill site has a slope favorable to the type of mill erected and is of sufficient area to allow of an economical arrangement of the various units of the plant. There is ample storage space for tailings and an excellent reservoir site available for the storage of water.

TYPE OF CONSTRUCTION

As the life of the property was estimated to be 17 years, or longer, depending upon new developments in the treatment of lower-grade ores, it was decided to make the various structures of a semi-permanent character. The buildings are of steel with corrugated steel coverings, except the concentrator which has a four-ply composition roof. All windows as well as skylights are of rubber glass. The floors throughout are of concrete, the retaining walls of reinforced concrete, and machinery foundations of massive concrete with little or no reinforcing. Reinforced concrete was used wherever applicable.

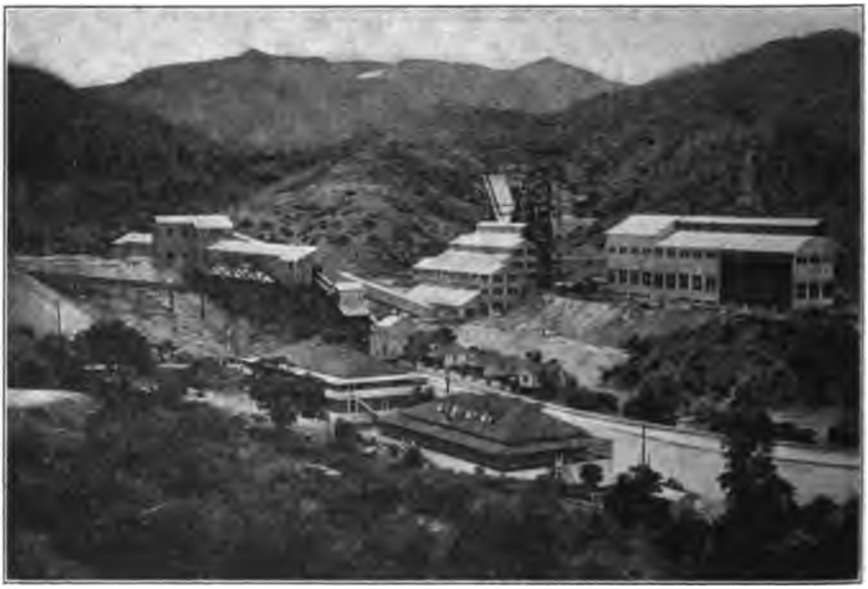


Fig. 1.—Mine Plant of Inspiration Consolidated Copper Co., Miami, Ariz. Looking West. Showing Compressor House, Main Shafts, Coarse-Crushing Plant and Storage Bins.

MINE PLANT

Under this general heading will be considered that portion of the plant with its adjuncts which takes the ore from the mine cars, hoists it to surface, reduces it to a size suitable for mill treatment and places it in storage, ready for transportation to the concentrator.

The relation between the orebodies and the mine plant can be described as follows: Conceive of a chain of orebodies the lateral extensions of which form a parabola with its vertex pointing to the north and having a length of approximately 9,000 ft. Place the main haulage ways on either side of the axis, 60 ft. apart, and open them to surface through two three-compartment shafts, 102 ft. centers, and about 750 ft. from the nearest approach of the orebodies. Group the mine plant symmetrically about the axis between the shafts and the orebodies and a general idea of their relation is obtained.

Dual Arrangement

As a result of the preliminary studies, the dual arrangement of the mine plant was evolved, the primary consideration being the necessity of handling 1,000 tons of ore per hour, which in itself precluded the use of a single shaft. Two shafts would also insure continuity of service. It was then considered advisable to follow this idea a step further and make the whole mine plant duplicate in its arrangement, which would give reasonable assurance against total shutdowns and also permit of a better load factor.

Underground Haulage

Underground haulage, for transporting crude ore from the stope chutes to the loading stations at the main shafts, is confined to two levels, the 600 or main haulage level, and the 400 level. The main haulage level is, in turn, made up of two distinct systems, each serving its main shaft. All haulage is to be controlled by a block signal system.

Two types of motive power for underground haulage were considered —electric and compressed air. On account of less danger to life due to the elimination of bare conductors, the compressed-air locomotive was chosen rather than the electric. The difference in the efficiency of the two systems is probably a small percentage of the total cost. "Safety First" was therefore the deciding factor.

The locomotives are of the two-stage, four-wheeled type, and have a weight on the drivers of 10 tons. The initial cylinder air pressure is 250 lb., and the charging pressure 800 lb. per square inch. The haulage capacity per charge was estimated at 50 ton-miles. The locomotives haul 25 cars, each of 5 tons capacity, on a 30-in. gage track over a 0.4 per cent. grade in favor of the load, and will negotiate a curve of 38 ft. radius.

All ore tapped from the stope chutes is broken to pass a 12-in. grizzly so that no further attention has to be given to it after it is once in the cars. At the shafts the cars, weighing 14,000 lb. each loaded, are dumped, five at a time, by motor-operated tipples, one for each shaft on the 600 or main haulage level, with a third on the 400 level. All hoisting is done from the main haulage level.

Tipples

The tipples have an overall length of 56 ft., and make a complete revolution in 15 sec. The driving shaft is connected through suitable gearing to a 35-hp. motor which runs continuously in one direction, the starting and stopping of the tipple being accomplished by means of a friction clutch located on the intermediate shaft. Each tipple is provided with an automatic stop which brings it into the proper position for running on and off the cars, this stop being released by means of a foot lever on the operator's platform. The two main tipples are operated from a centrally located platform.

Underground Pockets

The tipples on the 600 level dump directly into two main underground pockets (Fig. 2), which serve as storage for the four automatic measuring and loading devices that are placed beneath them. The capacity of each pocket is 1,600 tons, or 800 tons for each loader, which, with full pockets, will run the crushing plant about 3 hr. They are of reinforced-concrete construction throughout and are lined with 2-in. planks. The upper pocket has a capacity of 500 tons and is connected to the lower pockets by an inclined chute.

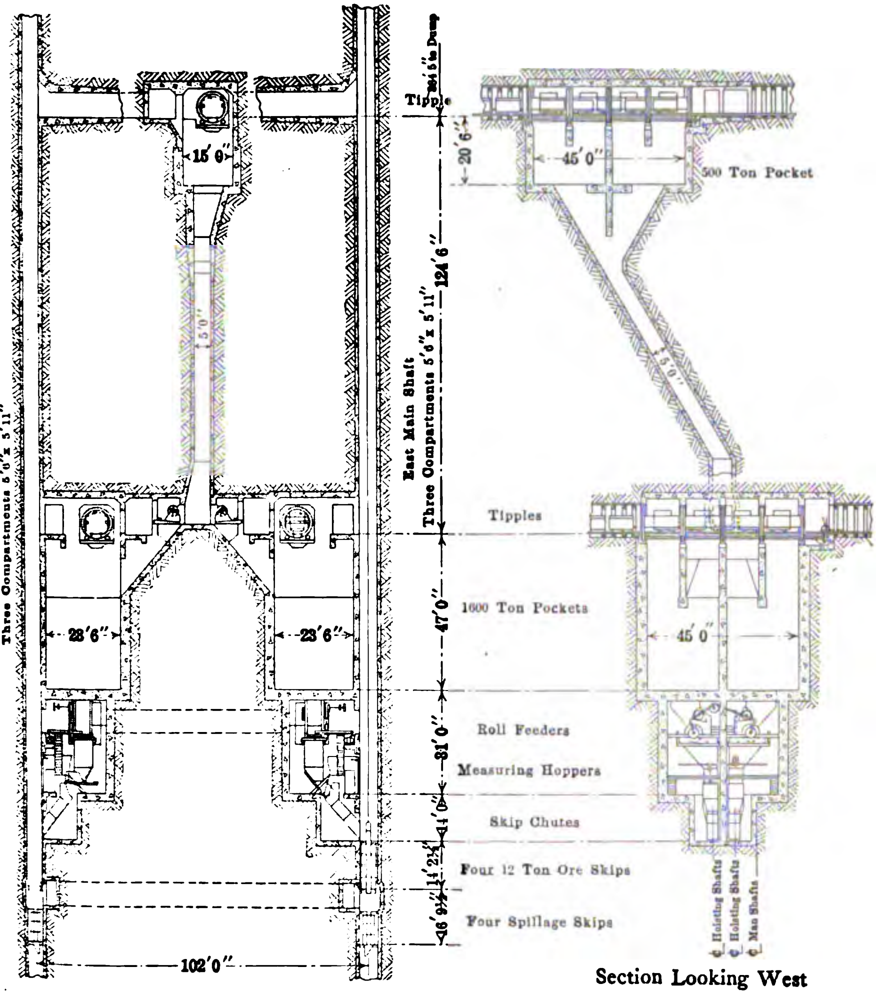
Automatic Measuring and Loading Devices

To carry out the automatic feature of the hoisting equipment, it was desirable to make the loading of the skips automatic. There being nothing on the market which could be used for this purpose, it was necessary to work up complete designs in accordance with original ideas.

In a few words the device (illustrated in Figs. 3*a* and 3*b*) can be described as follows:

Ore from one of the 1,600-ton underground pockets rests on a roll feeder which is actuated by a pawl and ratchet wheel, and is fed into a 12-ton measuring hopper. This hopper is suspended between the legs of a U-shaped scale beam with fulcrums at the two ends. Through a link at the middle of the yoke the proportional weight of the hopper and ore is transferred to the main scale beam which carries a counterweight of 1,370 lb. Motion obtained from the counterweight scale beam when the hopper becomes filled is utilized to operate a system of levers and lift the pawl from the ratchet wheel, thus stopping the roll feeders. The hopper is now ready to be emptied.

As the returning empty skip settles on the chairs, it strikes an arm projecting out into the shaft. This arm is connected by a system of links to the shaft on which the yoke of the gate forming the bottom of the hopper is hinged. The motion imparted to the system by the skip trips the gate yoke, the weight of the ore in the hopper causes the gate to drop into the skip chute, and the hopper is emptied into the skip. As soon as it is empty, a counterweight on the gate causes it to close, all motions are reversed and the measuring hopper is again filled.

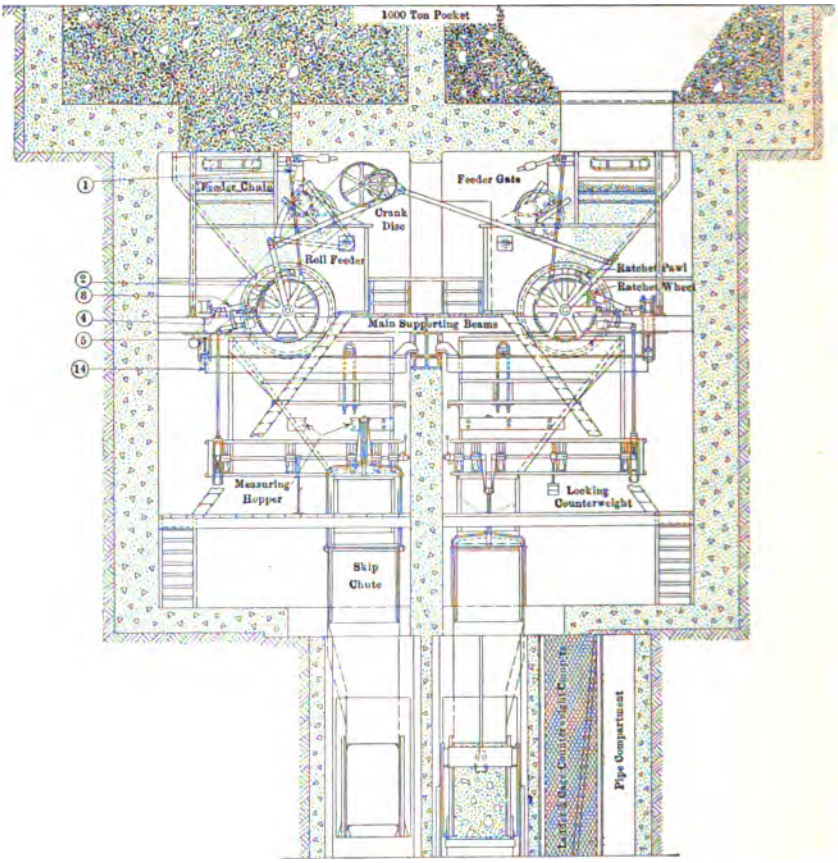


Section Looking North

Section Looking West

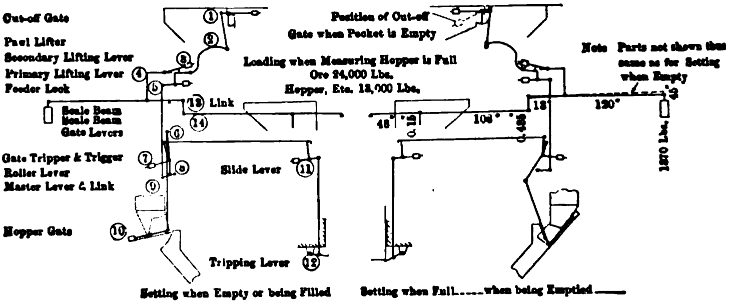
FIG. 2.—ARRANGEMENT OF UNDERGROUND POCKETS.

A cut-off gate prevents the roll feeder from becoming bare in case the pocket should be emptied. A 5-ft. layer of ore thus protects the feeder



Measuring Hopper Full Measuring Hopper being Emptied

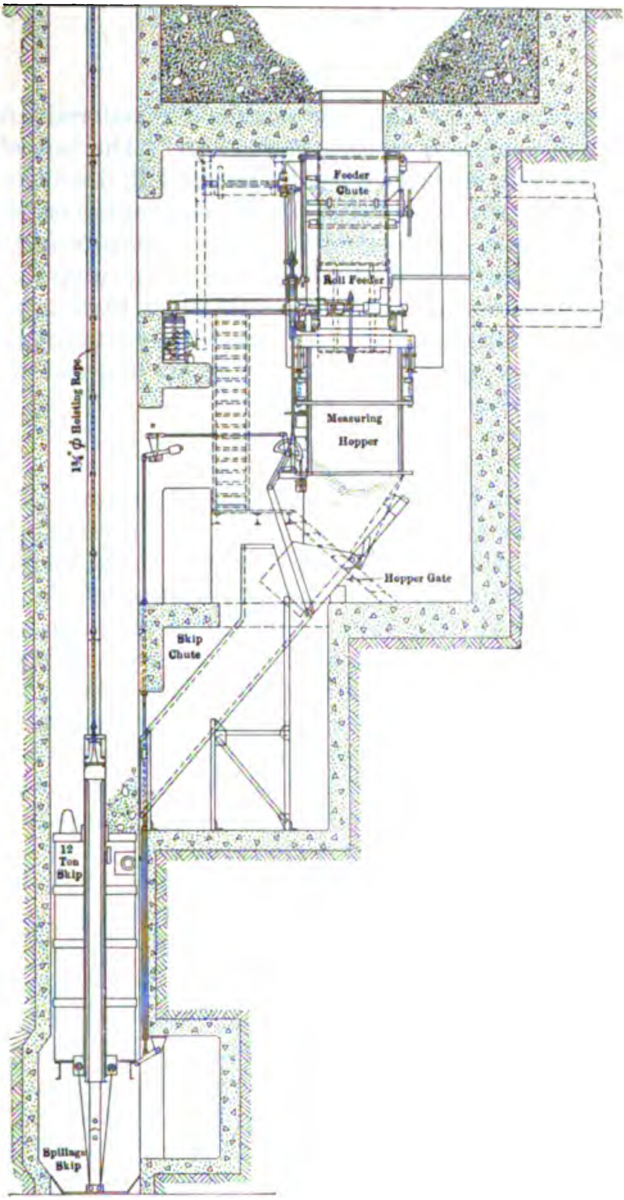
Sectional Front Elevation



Setting when Empty or being Filled Setting when Full----when being Emptied——

Line Diagram showing Principle of Operation

FIG. 3a.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.



Sectional Side Elevation

Measuring Hopper being Emptied Feed Cut-off

FIG. 3*b*.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.

and does away with excessive impacts due to the dumping of the ore from the tipple 50 ft. above. A 5-hp. motor furnishes the required power for the roll feeders of each pair of loaders.

Shafts

Taken individually the shafts do not deviate materially from usual designs, being of reinforced concrete throughout. They are of the same size, each being made up of three compartments 5 ft. 6 in. by 5 ft. 11 in., the two south compartments of either shaft being for the ore skips. The north compartment in the West Main serves the man elevator and the corresponding compartment in the East Main is taken up by pipe lines, ladderways and the elevator counterweight. All platforms, supports, and ladders in this compartment are of steel. The conduits carrying the power and lighting circuits underground are in the end walls, being brought to junction boxes every 100 ft.

Ore and Spillage Skips

The ore skips are built according to special designs. They have a nominal capacity of 12 tons and weigh 17,000 lb. each. The overall length is 21 ft., the skip proper being 14 ft. deep. The body is of 3/8-in. steel plate with liners to take up concentrated wear.

In order to keep the sumps at the bottom of the shafts clean and to facilitate the handling of ore which does find its way below the skips, spillage skips have been installed (Fig. 2). These rest on chairs 17 ft. below the ore-skip chairs and by means of hinged aprons completely close off the shaft. When one is filled it is attached to the bottom of the ore skip and hoisted to the top of the underground pocket, where by opening a hopper bottom it is discharged. They have a capacity of 9 tons each.

Automatic Hoists

At the time it was decided to sink two shafts the conditions seemed to favor the use of two entirely independent hoists, located symmetrically on either side of the main headframes. It was next proposed to put the two hoists under the same roof but to operate them independently of each other. Then on account of the advent of several new conditions it was deemed advisable to make the operation of the hoists automatic.

While the hoists are entirely automatic in their operation, either can be manually operated independently of the other if so desired. No attempt will be made to cover the automatic features of the installation as these are presented in detail in another paper.[1] Suffice it to say here that the problem as presented has been solved in a most efficient manner, sufficient proof being the entire satisfaction that the equipment has thus far given.

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[1] H. Kenyon Burch and M. A. Whiting: Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co., *Bulletin* No. 111, pp. 583 to 596. (March, 1916.)

Man Elevator

When the question of handling men came up, several possibilities as regards point of distribution presented themselves. They could be handled through an existing tunnel, hoisting or lowering to the working levels through an inclined shaft, or entirely through an inclined shaft, or through one of the main shafts. Whichever scheme proved the most feasible, the idea was to have the men do as little walking as possible, ample provision to be made for transferring them from change house to working places. It was finally decided that distribution from the main shafts was the most logical and the shafts were equipped accordingly.

The next consideration was the type of hoists to be used. Standard mine practice favored a separate hoist, manually operated and working against a counterweight. This type made necessary the services of two attendants, one on the cage and one at the hoist, but in working out probable operating costs the idea was conceived of eliminating the hoist attendant by an application of the elevator principle. Studies along this line were at once begun, but no great amount of enthusiasm among the manufacturers could be aroused on account of the unusual conditions. Equipment was wanted to handle a counterweighted, double-deck cage, weighing 7,100 lb. and loads of 7,500 lb., the maximum speed of the cage to be 800 ft. per minute. The fact that the hoist and counterweight would have to be removed from the elevator shaft also presented a new feature in the design. It may be said, however, that placing the hoist in the headframes directly above the shaft, as is the usual practice in office building design, was at one time considered.

The equipment as installed is operated entirely from the cage, the hoist, motor-generator set, and control apparatus being in the hoist house about 220 ft. away. The motor driving the direct-current generator on the motor-generator set has a rating of 190 hp., the generator developing 130 kw. The hoist is driven by a 158-hp. direct-current motor. It is provided with a safety brake, so arranged that when the hoist is stopped the brake is automatically applied to hold the cage. This brake is actuated by spring pressure, is constantly in service except when electrically released during normal operation of the hoist and is, therefore, instantly applied in case the current supply is interrupted from any cause. Additional safety devices are provided which take care of overwind, overload on motor, slack cable, etc. Two oil-cushioned buffers are provided at the bottom of the shaft, being so designed as to bring the loaded cage to a gradual stop in case the cage from any cause, while running at normal speed, should not stop at the lower terminal. The cage is equipped with a telephone which keeps the attendant in touch with the hoist house, also with an annunciator system covering the several levels.

Each level and the collar are provided with two-story stations that make it possible to load both decks without shifting. The cage accommodates 36 to 40 men besides the attendant. The surface station, which is in the headframe, is connected with the change house by a covered runway. From the underground stations the men are transferred to the various working places in cars accommodating 12 men each and drawn by the haulage locomotives.

The flow-sheet in Fig. 7 should be referred to in connection with the following description of the surface plant of the Inspiration company.

CRUSHING PLANT

The loaded ore skips assume a dumping position at a point about midway between the shaft collars and the center of the sheaves, leaving about 25 ft. for possible overwinding. The sheaves are 125 ft. above the shaft collars. They are 12 ft. in diameter and are pressed on 8½-in. shafts; 8-ft. sheaves on 6-in. shafts are used for the man-elevator ropes. The dumping tracks are of usual design.

Crude-ore Bin

By means of chutes leading away from one compartment of each shaft, an even distribution of the ore is effected throughout the length of the 2,000-ton crude-ore bin. This bin is of rather unusual design and very nearly self-cleaning. The bottom slopes down 45° each way from a hip center, toward the vertical back and front, a section resembling an inverted capital M. Gates are placed under the bin on the slope toward the back as well as on the front of the bin. This arrangement of bottom and gates not only allows the bin to be emptied but the ore is prevented from packing or hanging up by drawing off alternately from the rear and front gates. The gates are 48 in. wide and are operated by rack and pinion from runways.

For each of the four units of the crushing plant two gates, one under the bin and one in front, discharge on an apron feeder that travels across and under the bin, delivering the ore upon a 3-in. bar grizzly feeding a No. 8 gyratory crusher set to crush to 4-in. cubes (Fig. 4).

The apron feeders have a width of 48 in., a length of 25 ft. 6 in. between centers and travel at 7 ft. per minute; 20-in. belt conveyors traveling at 7 ft. per minute and driven from the feeder sprockets are placed beneath the feeders to take care of drippings. These discharge into chutes that join the undersize from the grizzlies. Wood scraps are removed from the ore at the discharge end of the apron feeders and also from the inclined conveyors leading to the disk crushers. These scraps are carried away by two 20-in. belt conveyors traveling at 75 ft. per minute, running across the building and dumping outside.

One of the four similar crushing units will be described in the following paragraph. Fig. 4 shows a cross-section of the crushing plant.

Description of Crushing Unit

The discharge from the gyratory and the undersize from the grizzly are conveyed by a 30-in. inclined belt conveyor to two 48-in. disk crushers, the ore first passing over a 1½-in. bar grizzly. This conveyor has a magnetic head pulley for removing tramp steel. The 48-in. disk crushers are set with a maximum opening of 2 in., their discharge and the undersize from the 1½-in. grizzlies joining on two 36-in. horizontal cross belts which in turn discharge at the center of the building on the two main

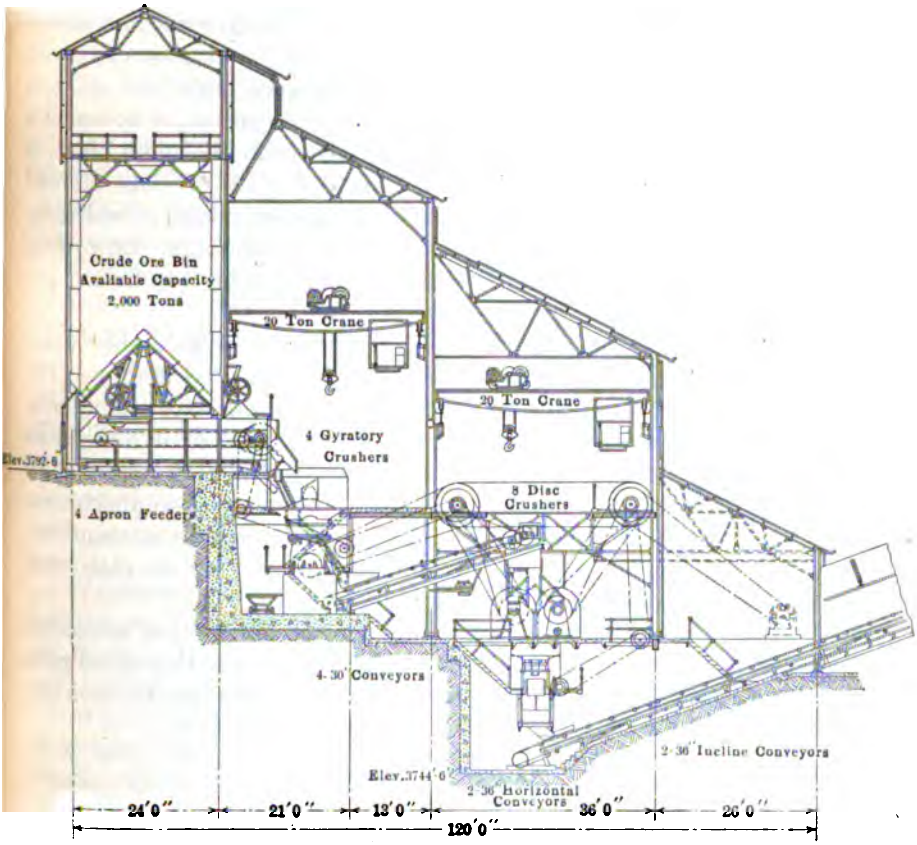

FIG. 4—SECTION OF COARSE-CRUSHING PLANT.

inclined belts, also 36 in. in width. These belts are 300 ft. long, travel at 350 ft. per minute and are equipped with hand-adjusted tail-pulley take-ups and also with weighted tension carriages. They discharge on four 24-in. horizontal conveyors over the storage bin, each of which has an automatic tripper which distributes the ore uniformly throughout the bin. Weightometers are placed on the two main inclined belts, thus giving a record of the ore handled.

STORAGE BIN

The purpose of the storage bin in connection with a mine plant is identical with that of a flywheel on an engine: "To give out and absorb energy when variation in the load occurs suddenly." It does away with the necessity of maintaining a nice balance between mine and mill and allows either, for a short time, to cease running altogether, or to run at a lessened capacity. Where the mill is removed from the mine plant, it also furnishes an economical means of transferring the ore into railroad cars. Besides this it allows of a hoisting and crushing period considerably less than full-time running, a very important consideration, since there are always repairs to be made in shafts, to loaders, tipples, hoists, and crushing machinery.

The Inspiration storage bin is double tracked, has an available capacity of 25,000 tons and will load a train of 14 cars without switching. The base is of reinforced concrete, and is provided with four expansion joints. The bin proper and conveyor housing are of steel. The main dimensions of the bin are as follows: Overall length 465 ft.; width 40 ft.; depth 40 ft. The bin is divided into three compartments, the two end compartments being for the storage of special ores if any such ever be encountered. They are comparatively small, accommodating one car on each track and having an available capacity of 1,700 tons each.

The distributing belts are driven by two 50-hp. motors located at either end of the bin. The ore is drawn off through hand-operated gates of the swinging cut-off type, there being three gate openings for each car.

SAMPLING PLANT

The heads sampler is placed at the point where the main inclined belts discharge on the horizontal cross belts (Fig. 7). This sampler consists of a rectangular cast-iron bucket, 9 by 30 in., on the end of a revolving arm 10 ft. in length. The bucket runs on a circular track 12 ft. 4 in. in diameter and is driven through a wormwheel mounted on a vertical shaft, making a revolution every 40 sec. The bucket has a swinging cut-off type of gate for a bottom. In its travel it cuts the two main ore streams, taking out $1/20$, and at a point 180° from the ore stream strikes a tripper which causes the gate to open and the sample to be discharged into a 10-ton hopper. From the hopper the sample is fed by a roll feeder and started on its way through the sampling plant, which is located against the center of the storage bin opposite the main conveyors. Here the sample is first passed through a 24-in. disk crusher. A $\frac{1}{5}$ cut is then made with a Snyder sampler and this cut is delivered to a set of 27 by 14-in. rolls. The roll product is cut by a second Snyder sampler, this $\frac{1}{5}$ cut going to a mixing drum that equalizes the flow. The resulting stream is delivered to a duplex Vezin sampler making two $\frac{1}{10}$ cuts. These samples are duplicates and weigh 110 lb. each, representing a day's run. The rejects from the three samplers go to an elevator that discharges into the storage bin.

MOTORS AND OTHER EQUIPMENT

The four units of the crushing plant are driven by four 200-hp. motors located in two dustproof rooms, symmetrically arranged on either side of the plant. Except for some minor reductions, belt transmission is used throughout.

Two 20-ton electric cranes are installed in the plant, one to serve the gyratories, the other the disk crushers, motors, etc. These are in turn connected to the yard track by transfer cars and a 20-ton electric trolley hoist, running at right angles outside of the building. Each 200-hp. motor is served by a 6-ton crawl so arranged as to allow of crane handling after leaving the motor room. A repair car and track near the gyratories permits of easy handling of their driving mechanisms, or bottom plates.

COMPRESSOR EQUIPMENT

Compressed air is used for two distinct purposes: For underground haulage, and for mine drills. For the first, 1,000-lb. air is used and for the second, 100-lb. The haulage system is served by two compressors, each having a capacity of 1,125 cu. ft. of free air per minute at 107 r.p.m. These are of the duplex, four-stage, power-driven type, the cylinders being cross-compounded, water-jacketed, and having intercoolers between the stages. These compressors have an automatic regulating arrangement whereby the 1,000-lb. discharge line is cut out when a predetermined high-pressure point has been reached, and the four cylinders converted into a double-compound compressor, discharging directly into the 100-lb. main. Each compressor is driven by a 430-hp. self-starting, synchronous motor.

The low-pressure equipment consists of two 100-lb. compressors having capacities of 3,000 and 7,270 cu. ft. of free air per minute. Both compressors are of the parallel, two-stage type, all cylinders being water-jacketed with intercoolers between the two stages. The 7,270-ft. machine is equipped throughout with Hoerbiger-Roegler plate valves, which on account of the very short movement and exceedingly light construction permit of a high compression efficiency. The cylinders of this machine are 46-in. and 28 by 36-in. stroke. It is driven by a self-starting synchronous motor of 1,150 hp. The 3,000-ft. machine is driven by a 500-hp. self-starting, synchronous motor at 107 r.p.m.

Two 100-kw., 125-volt, motor-generator sets arranged to operate in parallel serve as exciters to the compressor motors. Cooling water for the jackets and intercoolers is obtained from a circulating system consisting of a duplicate installation of 750-gal. triplex pumps and a spray cooling pond.

The compressors and hoisting equipment are housed in a spacious and well-lighted building which has a 25-ton electric crane running its entire length connecting at one end with a well into which railroad cars can be run and unloaded. A 22-panel, remote-control switchboard, together with the low-tension (2,200-volt) bus equipment, occupy a room along one side of the building, the high-tension transformer station being outside near one end of the building. Only two attendants per shift are required, one being an oiler. Without the automatic features for hoisting ore and the elevator for handling men, at least four would be required.

CONCENTRATOR

The considerations which led to the type of plant finally decided upon will first be given; this will be followed by a description of the plant. The concentrator includes the concentrates filter plant. Fig. 5 shows the concentrator with its adjuncts.

600-Ton Test Mill

The underlying principle that pervaded the work in this mill was to carry on all experiments with the idea constantly in mind that the particular method or machine under test might eventually be installed in the new mill.

The test mill was put up primarily to try out flotation on a large scale in order to determine its proper application to Inspiration ores. Since the flotation process, as applied to the concentration of copper ores, was an innovation, it was logical to suppose that in its last analysis the flowsheet might differ altogether from wet concentration methods. This did not prove to be the case, but the assumption was made and it was for this reason that the test mill was made so flexible.

The test mill was started February, 1914. The concentrates sold have paid the expenses of erection and operation, a remarkable record for a test plant. It has treated from 600 to 1,000 tons per day, the amount depending upon the types of machines being tested.

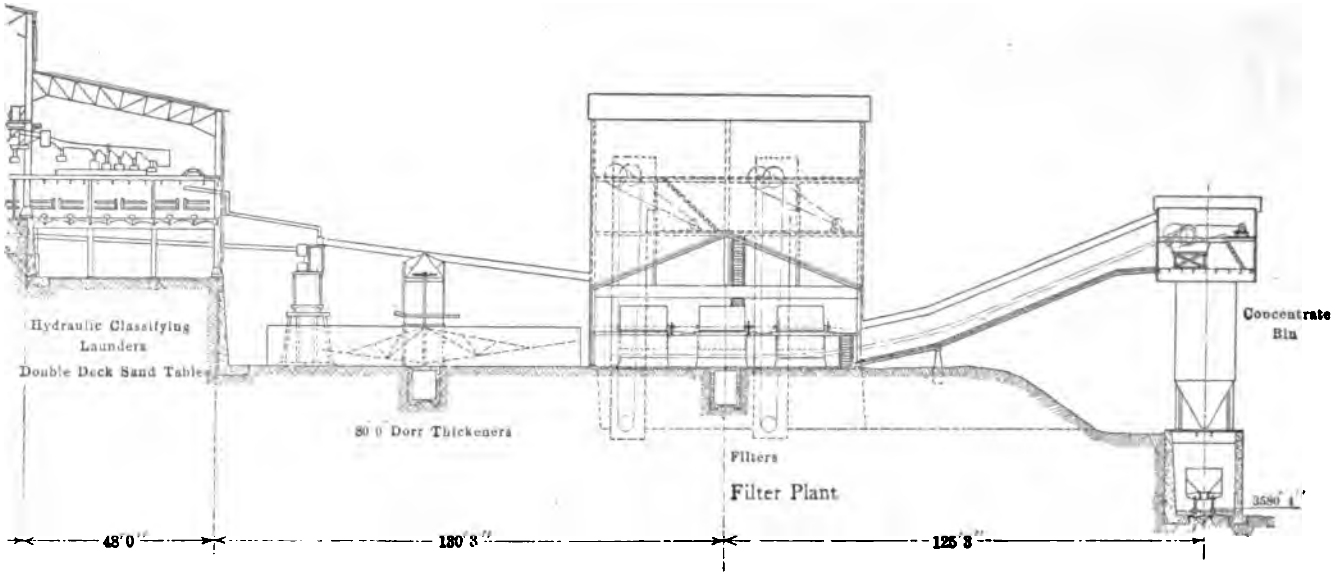

FIG. 5.—SECTION OF CONCENTRATOR AND FILTER PLANT.

The building is of timber, covered with corrugated steel. This type of building was chosen because it would facilitate changes or additions, was the cheapest and quickest form of construction, and would still have considerable value when dismantled. All machinery was placed on concrete foundations.

The ore for the test mill was taken from a development shaft and reduced to 3-in. cubes in a No. 8 gyratory crusher located nearby, the crushed ore being loaded into cars which emptied into a 180-ton bin at the head of the test mill. The following description gives the way in which the mill was first run, the various additions and changes being noted in sequence.

Two 30-in. apron feeders delivered ore from the bin to an inclined 20-in. belt conveyor, which was equipped with a recording weighing machine and a magnetic head pulley. This conveyor delivered the ore to a 36-in. Symons horizontal disk crusher and a Symons 48-in. fine reduction disk, or vertical disk crusher. These two machines delivered to a second 20-in. inclined conveyor. If so desired the two crushers could be bypassed, the delivery then being made direct to the second conveyor. This permitted the crushers below the second conveyor to be tried either as primary or secondary machines. The second conveyor delivered the ore to another 48-in. fine reduction disk and a Symons 48-in. roller mill. Water could be added either above or below these crushers. The ore then went to four Hardinge mills, the feed to which was regulated within suitable limits by a mechanical distributor. Three of these mills were 8 ft. in diameter with barrels 36 in., 44 in. and 72 in. long, all being direct-driven by induction motors. The fourth mill was 10 ft. in diameter, had a barrel length of 28 in. and was belt-driven. All of these machines were pebble mills with either silex or El Oro linings.

A drag classifier followed each mill, the whole floor being so arranged that each mill could be placed in a closed circuit with its drag or two mills and their drags run in tandem, that is, the first mill discharged into a drag, the sand from which was delivered into the second mill which in turn discharged into the second drag. The sand from the second drag was returned to join the sand from the first drag to form the feed for the second mill. The overflow from the drag was delivered either to eight sand tables by two distributors, or sent to other distributors which fed eight slime tables at the lower end of the mill.

Between the sand and the slime tables were placed the flotation machines which could be fed directly from the drag overflow or by any table product. The original installation consisted of two Minerals Separation eight-compartment flotation machines, one of 50-ton and the other of 600-ton capacity. By means of elevators any flotation product could be retreated on the slime tables or in the small flotation machine.

The concentrates from the tables and flotation machines went to a filter plant containing an Oliver and a Trent filter. The dried concentrates were loaded into 1-ton cars, weighed and dumped into a railroad bin from which they were taken to the smelter.

Mechanical samplers were installed at important points and hand samples taken where necessary.

The crushing and grinding experiments were not only extensive but proved most interesting, the results being quite unexpected. The 36-in. disk crusher represented the 48-in. disk crushers now installed in the crushing plant following the gyratories.

As secondary or intermediate crushers, that is, machines between the 36-in. Symons disk and the Hardinge mills the following were tried out: The Symons 48-in. fine reduction disk; the Symons 48-in. roller mill; the Symons 56 by 48-in. ring mill; the Bradley 66-in. centrifugal roller mill; the Overstrom centrifugal crusher; the Allis-Chalmers No. 4 hammer mill; and the Marcy ball mill.

The final results indicated that the Symons fine reduction disk, with a little redesigning, would be an efficient machine for the limited field of crushing from 4-in. cubes to 4-mesh. It was found that this could be done with a single pass, no oversize resulting. The machine would not work well wet and it was found necessary to remove all fines from the feed to the machine, otherwise a packing would occur. Consumption of steel wearing parts appeared low, as also did power consumption, although no definite results as to these two points were obtained.

The Marcy mill also proved to be a most excellent intermediate crusher, handling as high as 800 tons of 3-in. feed and under to pass an 8-mesh screen without return of oversize.

To compete with the four Hardinge mills a 6 by 20-ft. Chalmers & Williams tube mill and an 8 by 5-ft. ball mill were installed. The latter was a Marcy mill using 5-in. diameter steel balls. It was placed in a closed circuit, as were the pebble mills, and gave good results. It was next tried on a feed from the 36-in. disk crusher and then on a feed direct from the gyratory. These experiments definitely demonstrated that only three reductions were necessary: A gyratory, followed by a disk crusher and ball mills in a closed circuit. This is the arrangement installed in the present plant.

Several different classifiers were tried in the ball-mill circuit, a specially designed duplex Dorr classifier being adopted.

The flotation machines and the tests conducted on them are described in another paper,[2] but it is interesting to note here that the following machines were tried out: Minerals Separation, Hoover and Hebbard types; the Callow flotation cells; the Cole-Bergman; and the Inspiration machine.

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[2] Rudolph Gahl: History of the Flotation Process at Inspiration, *Bulletin* No. 117, September, 1916.

The Inspiration machine was devised during the operation of the test mill.

The concentrator is equipped as follows: 4 units with the Callow machines; 13 units with the Inspiration machines; and 1 unit with the Minerals Separation machines.

The drag classifier as originally tried out proved to be too small for the tonnage required, the drag now in the concentrator having been developed from the small one.

The tables tested were the Deister Machine Co.'s double-deck slimers, double-deck sand tables, and four-deck slimer; the Deister Concentrating Co.'s single-deck slimer, double-deck sand table and No. 4 sand machine; and the Wilfley No. 6 table with decks arranged for both sand and slime. The Deister Machine Co.'s double-deck sand table was adopted, 198 being installed.

This sketch of the test mill does not bring out the many small things experimented with, such as shaking screens, Caldecott cones, samplers, feeders, slopes of launders and conveyors, or the necessary efforts to work out improvements on the new machines.

THE 15,000-TON CONCENTRATOR

The ore comes from the mine plant in 14-car trains, and is dumped into the concentrator bins, which have a capacity of 12,000 tons, or 670 tons per unit (Figs. 5 and 6). These bins are of the suspension type, are double tracked, and extend the full length of the mill, 300 ft.

The mill is made up of 18 units, similar in equipment except for the differences in the flotation department. Eighteen 30-in. apron feeders, located on the center line of the bin, feed the ore to 20-in. inclined conveyor belts traveling at 150 ft. per minute. Weightometers are installed on these belts, which record the amount of ore fed to each unit. The conveyors discharge into hoppers where a split is made, half of the feed going to the north ball mills and half to the south mills, the transfer being made through launders with cast-iron liners and having a slope of 45° (Fig. 7).

Grinding Section

Floor Arrangement.—One of the novel features relative to the layout of this plant is the arrangement of the grinding floor. With due regard for the individual capabilities of the machines used, it is nevertheless a fact that the arrangement has been important in bringing about the excellent results obtained. The following tabulation will serve to show that the method, besides being efficient in the work it is doing, is conservative of floor space as well:

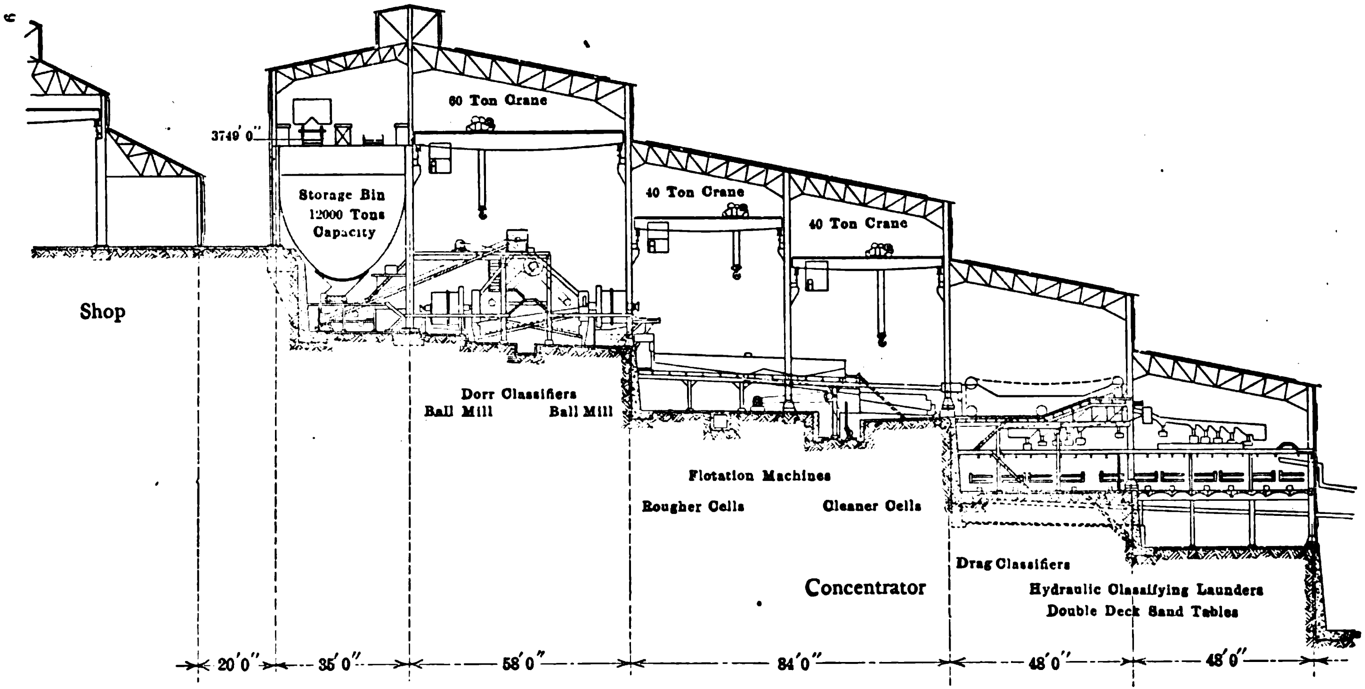


FIG. 6.—SECTION OF CONCENTRATOR.

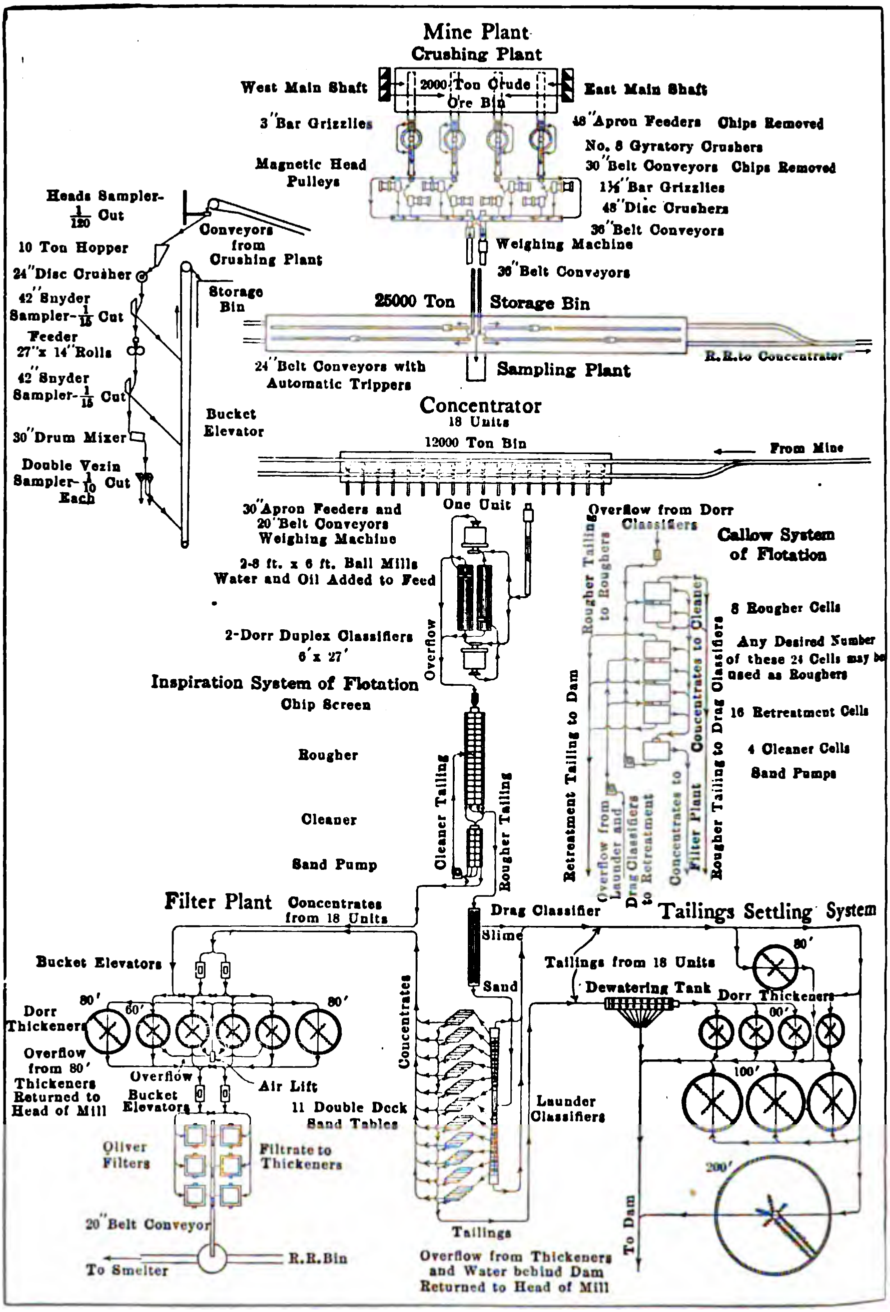


FIG 7.—FLOW SHEET OF INSPIRATION PLANT.

Capacity of installation, 15,000 tons per day from 2-in. disk crusher opening to 1½ per cent. on 48-mesh.

Area of grinding floor, exclusive of motor platforms, 300 by 66 ft., or 19,800 sq. ft.

Floor space per ton capacity, 1.32 sq. ft.

The scheme can be briefly described as a parallel-series arrangement; two circuits—each with a Marcy ball mill and a Dorr classifier in series—are arranged in parallel, the oversize from each classifier being returned to the mill opposite for regrinding. The mills are symmetrically arranged in two rows extending the length of the mill with their feed ends facing each other, just enough room being left between them to accommodate the Dorr classifiers (Figs. 10 and 11).

The Dorr classifiers act as elevators and since all return is handled by them, the whole department is confined to a single level. There is a main runway between the mills above the classifiers, from which branch runways are taken off to lead to the motors, between mills, etc. This arrangement gives the attendant an excellent opportunity to watch his machines and to reach them when necessary. The conveyors bringing the feed to the mills are terminated in hoppers on a platform high enough above the mills to allow of a 45° slope for the launders leading to the feed boxes. The motors for driving the conveyors, classifiers, etc., together with their control apparatus, are located on this platform, as are also the control panels for the ball-mill motors. The switchboard attendant thus has a commanding view of the whole grinding floor. The floor is served by a 60-ton electric traveling crane which transfers the mills to and from the repair floors located at either end of the building. An inclined skipway along one side of the building serves the whole concentrator, the cranes on each floor making direct connection with it.

Marcy Ball Mills.—The feed for each ball mill consists of ore from the bin, together with the sand from the discharge of the mill opposite, the final product of the system being the overflow of the classifiers. The feed to the ball mills has a consistency of about 1 to 2, and the overflow from the classifiers about 3 to 1, the latter being maintained by an automatic device which adds water as required in the discharge boxes of the ball mills.

The ball mills are 8 ft. in diameter and 6 ft. long. They are direct-driven through herring-bone gears by 225-hp. induction motors. Our experiments with the ball mill in the test plant led us to the conclusion that we could crush 400 tons of the coarse-crushing plant feed with each mill in 24 hr. to 2 per cent. on 48-mesh screen and 60 per cent. minus 200. Now that the Inspiration mill is running with all sections, it has been shown that our test figures were correct. Not only have we been able to crush 400 tons per mill, but with 18 sections running the average daily dry tonnage for the month of May was 15,358. We have a test section (of two mills) that showed for a short period a tonnage rate of 1,000 tons per 24 hr. While this is by no means the average, yet it indicates what may be done in the future.

Performance of Crushing Machines.—In view of the character of the ore and the fact that much of it is tough and more difficult to crush than the ordinary porphyry ores, we consider the power consumption for our crushing plants very satisfactory. The power consumption at the coarse-crushing plant for the month of May, taking mine-run ore, crushing it in gyratories and disk crushers, and conveying to the railroad bins, was 0.329 kw.-hr. per ton. At the mill, the average power consumed per ton during the month of May for the ball mills, including power for feeders, conveyors and classifiers, was 10.4 kw.-hr. Therefore, the total power consumed per ton for the month of May on ore taken from the mine bins, crushed and delivered to the flotation machines was 10.729. We have had one section of the ball mills in operation for a period of 10 days that has given a power consumption of 8.53 kw.-hr. per ton. If we add to this figure the power used at the coarse-crushing plant and of the conveyors, we would have 8.88 kw.-hr. per ton of mine-run feed delivered to the flotation machines. The above does not include the power required for transporting the ore by steam road from the mine to the mill.

The crushing and delivery of the ore for concentration is a matter of great importance, and one that absorbs a large portion of the operating costs in the treatment of ores. Too much attention cannot be given it. We are now conducting many experiments in connection with our crushing problem, including different types of steel for wearing parts and modifications that influence our tonnage and cost of production. We have equipped ball mills without the grates and the results of the mills so equipped have shown us that ball mills equipped with grates have advantages in increased tonnage and efficiency. Our experiments with the different shapes of pebble mills, our subsequent experiments with the ball mills, and finally our monthly operation, lead us to the conclusion that a good choice was made in the selection of our present grinding machinery.

Flotation Section

The overflow from the Dorr classifiers goes to the flotation machines, of which there are three types now in operation. Generally speaking, the flotation machines make a clean concentrate which goes to the filter plant, a middling which goes to drag classifiers and a tailing which goes to waste. As the subject of flotation is to be taken up in detail in another paper, just enough will be given here to bring out the part the process plays in the general scheme. The flow-sheets for the two processes differing somewhat, each will be considered separately.

Flow-Sheet with Callow Flotation Cells.—A unit installation of these machines consists of a total of 28 cells which for sake of compactness are grouped into sets of four cells each, individual cells being 3 ft. 3½ in. by 10 ft. 2 in. Of this total eight are roughers, 16 are for retreatment and four are cleaners, the arrangement of the group being such that all feeds and products except two are handled by gravity. These two are elevated by sand pumps. The Callow cells, as installed in this plant, make two products, a concentrate and a tailing.

The overflow from the ball-mill classifiers first undergoes a roughing treatment in the eight roughers. Here the two products are a concentrate which goes to the four cleaners and a tailing which goes to a large, specially designed drag classifier, a detailed description of which will be given later. The concentrate from the cleaners is a finished product and goes to the filter plant. The tailing from these cells is pumped back into the rougher cells for a second passage through the system. The tailing from the rougher cells is classified in the drag into two products, a sand and a slime overflow, the sand being still further prepared for table concentration in hydraulic classifiers. The tables make two products, the concentrates joining the flotation concentrates in the filter plant, and the tailing going to waste. Further treatment of a middling product from these tables is contemplated and studies have been made along this line, but as yet nothing definite has been reached. The overflows from both the drag and the hydraulic classifiers are united and pumped back to the 16 re-treatment cells, the concentrate from which goes to the cleaners, and the tailing to waste. Throughout the mill adequate provision has been made for sampling.

Flow-Sheet with the Inspiration Flotation Machines.—Except for a special arrangement in one unit, the only difference between this and the Callow flow-sheet is in the flotation department.

From a metallurgical standpoint the difference here is not radical. In a few words the essential difference can be stated as follows:

The Inspiration system makes use of but two machines per unit, a rougher and a cleaner placed end to end (Fig. 9). These are each designed to allow a free passage of the feed from the upper or feed end, to the tailing or discharge end, during which passage the feed is subjected to the frothing action of several compartments. Because of the large number of compartments (16 for the rougher), it has been found that a tailing can be made, the slime of which does not require re-treatment as is the case in the Callow system. The sand contained in the rougher tailing is subjected to table concentration. The tailing from the cleaners is pumped back to the roughers for a second treatment.

Several mechanical features have been introduced which are expected to keep the cost of operation at a minimum. The arrangement is simple,

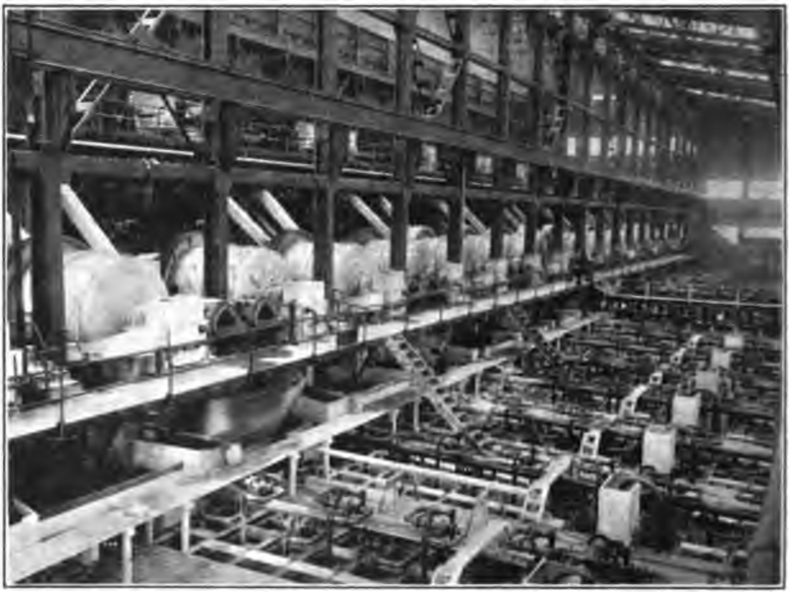


FIG. 8.—BALL MILLS AND FLOTATION MACHINES.

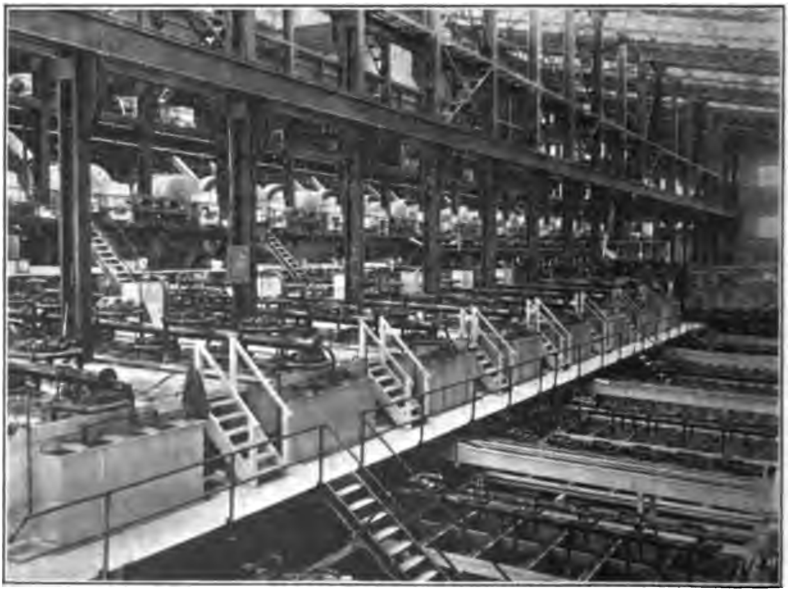


FIG. 9.—INSPIRATION FLOTATION MACHINES.

requires a minimum of floor space and is easily attended. Its large capacity is an important feature.

Flotation Air Supply.—The flotation department requires a large volume of low-pressure air. This is supplied by four single-stage centrifugal compressors occupying one end of the flotation floor. The rating of these compressors is as follows:

Capacity, 23,000 cu. ft. of inlet air per minute.
Discharge pressure, 5.75 lb. per square inch.
Revolutions per minute of impellers, 3,850.
Horsepower of driving motor, 720.

The step-up to the impellers is made through Alquist gears. One unit of the four is held in reserve.

Concentrates Filter Plant

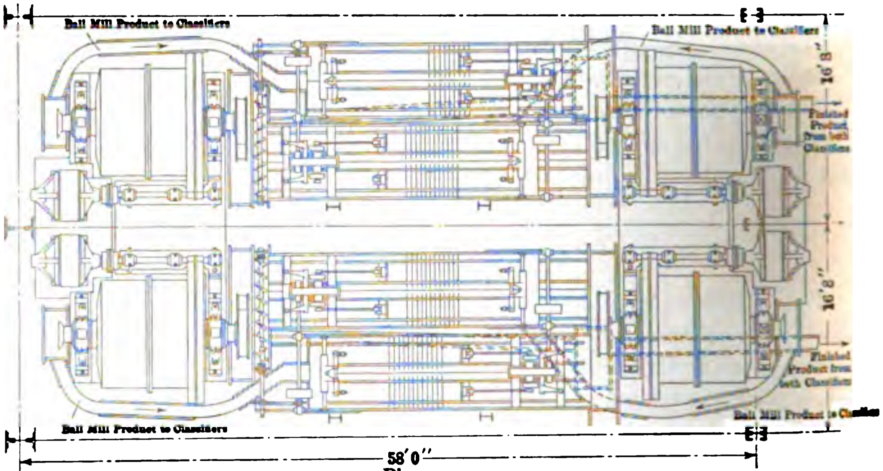
Both the flotation and table concentrates receive the same treatment. From the concentrator they are either elevated or flow by gravity into the five 60-ft. and three 80-ft. Dorr tanks, located on either side of the elevator house. Here the moisture content is reduced to about 50 per cent. The thickened product is drawn off through the bottoms into launders and conveyed through tunnels to bucket elevators, which raise it to the top of the elevator house where it is distributed to six Oliver filters (Fig. 5).

The overflow from the Dorr tanks is practically clear and flows to the return-water sump. The filters are standard Oliver filters, the drums being 11 ft. 6 in. in diameter and having a length of 12 ft. After leaving the filters the concentrates have a moisture content of about 17 per cent. The arrangement is such that a single 20-in. conveyor belt traveling at 100 ft. per minute takes the concentrates away from the filters and delivers them to a railroad bin of the tank type. Here they are loaded into hopper-bottomed steel cars of 60 tons capacity. A 350-ft. train shed protects the loaded cars from heavy rains and winds.

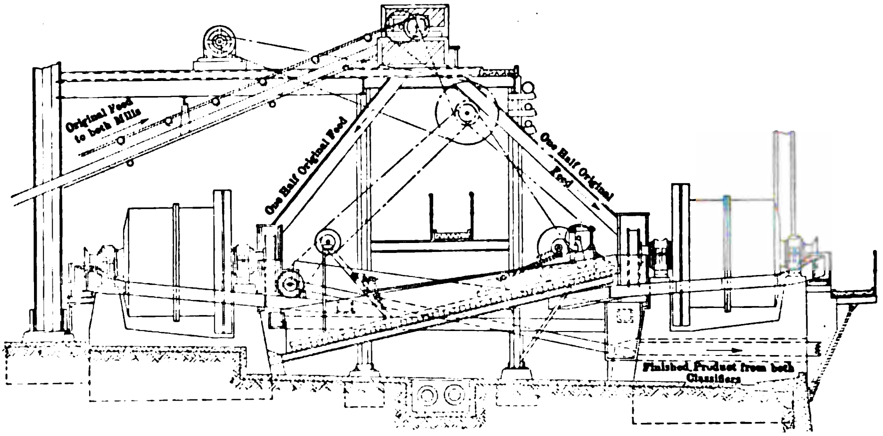
Drag Classifiers

The drag classifier used on the flotation tailings for separating sand from slime is the result of experimental work in the test mill. It is worthy of a detailed description not alone on account of its ability as a classifier but also because of its large capacity. Except for the concentrates taken out by the flotation cells, each machine handles the entire tonnage of one unit, or 800 tons per day.

The classifier consists essentially of two 18-in. belts, running parallel to each other and so arranged that the return belts run entirely clear of the settling surface, the rough classification being carried on sufficiently far below the surface to cause no disturbing currents. This is made possible by the use of four pulleys, one at the discharge end, two (one vertically above the other) at the feed end, and one at the center near the bottom, where the slope begins. The distance between head and tail pulleys is 39 ft. The direction of travel of the return belts is horizontal, but on passing over the upper tail pulley it becomes vertical. The belt thus enters the pulp perpendicular to its surface. At the bottom of the tank its direction is again changed to the horizontal by the lower tail pulley. It travels thus to the center of the tank where a break is made; the belt with its load passes under another pulley and starts up a 20° slope toward the discharge end. All pulleys are 32 in. in diameter.


Ball Mill Product to Classifiers
Ball Mill Product to Classifiers
16'8"
Finished Product from both Classifiers
16'8"
Finished Product from both Classifiers
Ball Mill Product to Classifiers
Ball Mill Product to Classifiers
58'0"

Plan
Runways Omitted


Original Feed to both Mills
One Half Original Feed
One Half Original Feed
Finished Product from both Classifiers

Elevation

FIG. 10.—BALL MILL AND CLASSIFIER ARRANGEMENT.

Two adjustable overflow launders on either side of the tank having a combined lip length of 52 ft., carry off the slime. The effective settling area of the tank is 25 ft. by 4 ft., or 100 sq. ft. The overflow is 5 ft. above the bottom horizontal belt. The two sets of submerged pulleys are mounted on shafts running in water-tight bearings supported by the sides of the tank.

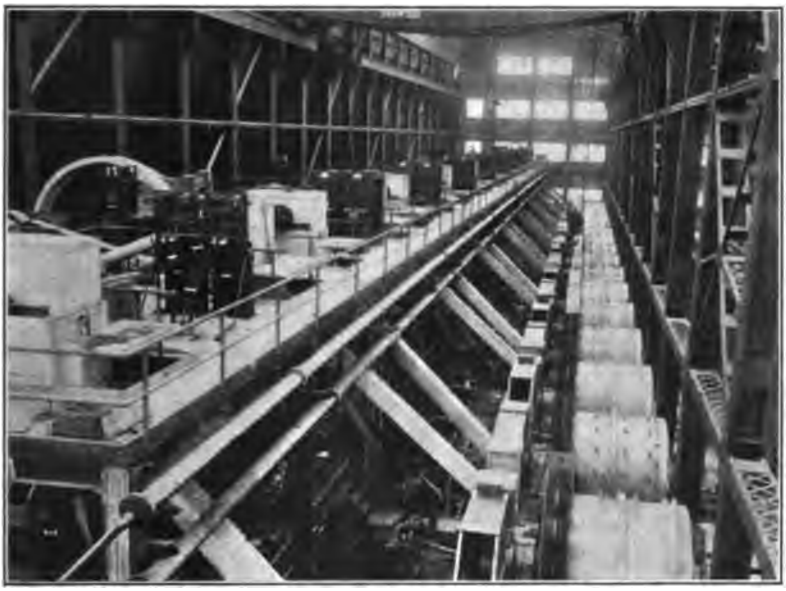


FIG. 11.—BALL MILLS AND MOTOR PLATFORMS.

Tailing Settling System

The conservation of the water supply is imperative, and it was for this reason that the extensive tailing-settling and return-water system was installed. The water is reclaimed at the concentrator and at the tailing dams. The equipment at the concentrator consists of three 100-ft. and one 200-ft. Dorr thickeners, and a large dewatering box 17 ft. by 109 ft. The overflow from the 80-ft. concentrate tanks is also turned into the return system. The dewatering box handles only the table tailing, the overflow from it being further settled in three of the 60-ft. Dorr thickeners. The overflow from all tanks goes to a 210,000-gal. concrete sump from which it is pumped to a 60-ft. diameter steel tank of like capacity on the shop level above the concentrator.

The return-water pump equipment consists of four vertical triplex pumps direct-driven through gearing by 100-hp. synchronous motors. These pumps each have a capacity of 2,000 gal. per minute and work against a head of 113 ft.

The water reclaimed from behind the dam (Fig. 12) is at present being handled by a 3,000-gal. two-stage centrifugal pump, so arranged that it can be moved up a skidway as the water level of the pond behind the dam rises.

200-Ft. Dorr Thickener.—The 200-ft. Dorr thickener is larger than any single unit ever installed and is, therefore, interesting from a mechanical standpoint. The thickener has not yet been erected but at this time the designs are complete. It consists essentially of a reinforced-concrete tank 200 ft. in diameter with the bottom sloping toward the center and having a depth at the outside of 7 ft. 3 in. The feed is delivered at the center of the tank through a launder supported by a 107-ft. steel truss. At the center of the tank is a steel plate pivot, 18 ft. high, upon which is mounted the drive drum. To this drum are fastened four short rakes (two 24 ft., and two 35 ft.) and the driving truss, which in turn carries two long rakes, 10 ft. apart and 100 ft. in length. The slope of the bottom of the tank varies, it being 2$\frac{3}{16}$ in. per foot near the center, 1$\frac{3}{4}$ in. per foot for the short rakes and the inner 16 ft. of the long rakes, and the remainder 1 in. per foot. The driving truss is completely submerged except for the outer end, which rests on a circular track on the tank wall. The driving mechanism is placed at the outer end and consists of a 5-hp. motor which is connected through gearing to a single 24-in. plain-tread driver. Two similar wheels are placed 10 ft. on either side of the driver to distribute the weight. The weight of the driving truss provides the necessary traction. The thickened product is drawn off through an

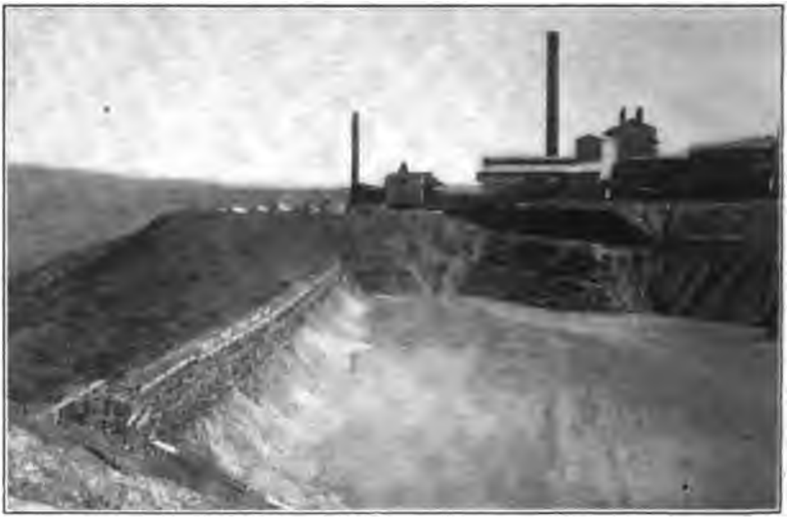


FIG. 12.—TAILINGS DAM.

annular opening at the center; and the overflow at two diametrically opposite points on the periphery of the tank.

GENERAL EQUIPMENT AND SUPPLIES

Water Supply

Various possibilities were presented for supplying water to this plant. To insure a supply, sufficient ground and water rights were early purchased at Wheatfields, about 12 miles from the mill site and 1,000 ft. lower in elevation. Before developing this supply, however, it was decided to prospect on the flat below the tailing storage site. Wells were sunk at various points until the best location had been determined, which is 2½ miles from the mill, 430 ft. lower and at the junction of two fair-sized drainage channels, receiving their supply from the Pinal Mountains about 10 miles away and 4,000 ft. higher. Here six wells have been sunk and 24-in. multi-stage turbine well pumps installed. These deliver through wooden pipe lines to a common steel sump tank having a capacity of 235,000 gal. Each well pump is belt-driven by a 150-hp. vertical motor and delivers a maximum of 1,200 gal. per minute. Near the sump tank is located the pumping station which contains six 1,200-gal. pumps delivering into a common 20-in. pipe line. These pumps are horizontal, duplex, double-acting and are direct-driven through herringbone gears by 300-hp. synchronous motors taking current at 6,600 volts. Power for the pumping station and wells is supplied by the Inspiration-International power house. A 10-ton crane serves all parts of the building.

The 20-in. pipe line is 14,600 ft. long with a rise of 520 ft., and delivers water to the storage reservoir near the concentrator. From this reservoir water is delivered to all parts of the property. It is located about 80 ft. above the concentrator, 1,200 ft. away and has a capacity of 3,000,000 gal. An oval excavation about 20 ft. deep was made in the top of a hill and the sloping sides and bottom lined with concrete. Two 14-in. pipe lines deliver the water to the concentrator.

Power Supply

Electric power is obtained from two sources. The Reclamation Service of the United States Government, from its hydro-electric plant at the Roosevelt Dam, 40 miles away, furnishes energy at 40,000 volts, 25 cycles, three-phase.

In conjunction with the International Smelting Co., the Inspiration company built a power house that utilizes the waste heat from the reverberatories under one set of boilers. Another set of boilers, oil-fired, is also used when the power demand is greater than can be supplied by the waste-heat boilers alone. This power house contains three 7,500-kva.

steam-turbine-driven generators, delivering energy at 6,600 volts, 25 cycles, three-phase; also three 15-lb. reciprocating blowing engines, each having a capacity of 15,000 cu. ft. of free air per minute. The air is for the smelter use exclusively, while the electric energy is for the mining, smelting and ore-dressing plants.

The distributing system is so laid out that any division of the plant can take power from either source. When available, the Reclamation power is used for the greater part of the Inspiration load. Each unit of the plant has its transformer station for stepping down the transmission line voltage.

Motors of 50 hp. or greater are run on 2,200 volts, smaller motors on 440 volts, and lighting circuits on 110 volts. Synchronous motors are used as required to maintain a satisfactory power factor.

TRANSFORMER STATIONS

There are two separate outdoor transformer stations, one at the concentrator and one at the mine plant. These stations are of rather unusual design, being skeleton-steel structures. The supports are of pipe poles and the trusses between which the busses are stretched are angle lattice construction.

As the power from the two sources is received at different voltages each station requires two sets of transformers. The Reclamation energy is delivered at 40,000 volts and the power house at 6,600 volts, both being stepped down to 2,200 volts for distribution about the plant.

Both high-tension transmission lines are in duplicate circuits. The incoming lines pass over electrolytic lightning arresters to electrically operated remote-controlled circuit-breakers, then through electrically operated remote-controlled automatic circuit-breakers to the transformers. Any one transformer may be cut in or out as desired, or all at one station can work in parallel. At the concentrator there are eight outdoor-type, 2,000-kva., oil-insulated, water-cooled, three-phase transformers, four on the 40,000-volt line and four on the 6,600-volt line. At the mine there are four similar transformers, two for each incoming line.

The 2,200-volt circuits are led from the transformers to the distriuting stations, the one at the mine being located in the compressor and hoist house and the one at the concentrator in a separate building. Each distributing station consists of a concrete cell structure, carrying two sets of 2,200-volt busbars, one for each power line. This cell structure also carries the disconnecting switches, meter transformers and oil switches as required by the remote-control feature of the switchboard. In front of the cells is the switchboard with separate panels for the control of each incoming high-tension line, each set of transformers and each outgoing 2,200-volt circuit. Wattmeters, frequency meters, and power-factor meters, all graphic, and integrating watt-hour meters are installed for each power supply, and ammeters, watt-hour meters and graphic wattmeters for each outgoing circuit. Two-throw oil switches, controlled from the switchboard, make it possible to run any circuit on either power, or to disconnect it from both.

The transformer and distributing stations, besides presenting an attractive appearance, are unusually complete, convenient for the attendant and provide the greatest possible safety.

SHOPS AND RAILROAD FACILITIES

All shop work, except that for the mine, is done at the concentrator shops where a complete equipment is installed. The equipment comprises about 30 different machine tools, all with individual motor drives, a 40-ton electric crane, traveling the entire length of the building, the necessary facilities for locomotive repairs, several welding outfits for various classes of work and a forge shop equipped with both oil and coal forges. The warehouse and electric shop and the locomotive roundhouse occupy opposite ends of the shop building. A standard-gage track through the center of the building permits cars to be run either into the shop or through the shops into the warehouse.

The shop level is 22 ft. above the upper floor of the concentrator. Communication between the two buildings is provided by the inclined skipway which occupies one end of the concentrator. This connects with the shops by a well through which material can be transferred either into the shops from the mill for repairs, or into the mill from railroad cars.

The railroad facilities comprise the main line and several spurs, the total length being about 10 miles. All equipment is standard gage. The lines connect the several units of the surface plant, except the pumping plant and the Live Oak Division. These are reached via the Arizona Eastern. On account of the topographical conditions, it was necessary to use switchbacks to make the grade between the Arizona Eastern on the flat and the concentrator, about 400 ft. higher. The switchbacks provide for handling six cars. The extremes of grade and curvature on this line are 4 per cent. and 12° respectively. The line which connects the mine and the mill has a grade of 0.3 per cent. in favor of the load.

The six-wheel, side-tank type of locomotive is used. Two are used on the mine line. These have a weight on the drivers when loaded of 112,000 lb. A third locomotive is used on the low line, which has a corresponding weight of 200,000 lb.; the latter being equipped with a superheater and Walschaerts' valve gears, oil burners being used on all. The ore cars are of the Ingoldsby patent-dump type, have a capacity of 60 tons and a train length of 33 ft. They are of all-steel construction and weigh about 22 tons each.

Electric haulage on the mine division was at one time considered, but for sake of uniformity on the several lines the idea was abandoned.

CONCLUSION

The salient features of the mine plant can be summarized as follows: The dual arrangement of the entire plant; the automatic loading and hoisting of ore; the application of the elevator principle for handling men; and the reduction of the crude ore for mill treatment in two simple stages.

The final product of the plant is ore in storage ready for mill treatment. It carries roughly 1¾ per cent. copper, mainly sulphides, and has been crushed to pass a 2-in. opening of the disk crushers. The average moisture content is 2½ per cent.

To the mine summary can be added the following covering the remainder of the plant: A present mill recovery on sulphides of 90 per cent., or 85 per cent. on the total content; finished grinding in one machine; and flotation followed by table concentration.

Although the Inspiration plant is not quite the largest, it does embody what is today the latest practice in the treatment of low-grade chalcocite ores. In certain departments considerable pioneering has been done, which has resulted in marked economies, advantage of which will no doubt be taken where similar conditions make them applicable.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.

BY GEORGE R. LEHMAN, B. A., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

THE Inspiration Consolidated Copper Co. had an orebody at Miami, Ariz., of close to 100,000,000 tons of low-grade copper ore, and the method of mining this ore most profitably was of great importance. The selection of a method was practically limited to a "top-slicing," a "shrinkage-stope" or a "caving" method. The "top-slicing" method would give a high extraction of the developed ore, at a high mining cost, and the "shrinkage-stope" or "caving" methods a lower extraction of the developed ore, at a lower mining cost.

ORE-DRAWING TESTS

To decide on the method to be used it was necessary to know about what extraction of the ore could be obtained by the "shrinkage-stope" or "caving" methods. As no definite information regarding the question was available at the time, C. E. Mills, the general manager, thought that some information could be obtained by experimenting in drawing ore, covered with capping, from an experimental box. Some work had already been done along these lines by W. C. Browning for the Inspiration Copper Co. and by myself for the Live Oak Development Co.

The idea was to represent as nearly as possible, in the experimental box, the conditions within an area of caved stopes of caved ore all ready for drawing. That is, the area was supposed to represent a semibroken mass with the capping above it in the same condition and ready to follow the ore downward as it was drawn out of the chutes under the ore. To represent the above condition, in the experimental box, crushed ore from the mine was placed in it to a given height, and red barren capping from the mine was placed on top of the ore.

The Experimental Box

The experimental box shown in Fig. 1 was made of wood and glass. The glass was used for the sides so that the drawing action of the outside row of chutes could be seen and noted, the latter being cut in the center by the glass sides. The length of the box was 30 in., width 20 in. and the height 25 in., and built to a scale of 1 in. = 5 ft., the box represented an area of 150 by 100 ft. and a height of 125 ft. within the mine. The box, as shown in Fig. 1, was divided in the center and only half was used for each test. The bottom was bored with ½-in. holes, representing chutes, 1¼ in. center to center, thus indicating chutes of 2½ ft. diameter spaced on 6¼-ft. centers according to the scale. Thus the ore could be represented as drawn at either 6¼, 8⅞, or 12½-ft. chute intervals. The holes were fitted with wooden plugs through which wire nails were driven, so that when the plugs were in place the heads of the nails projected just the least bit above the inner bottom of the box. The nail was used to clear the chute when it "hung up."

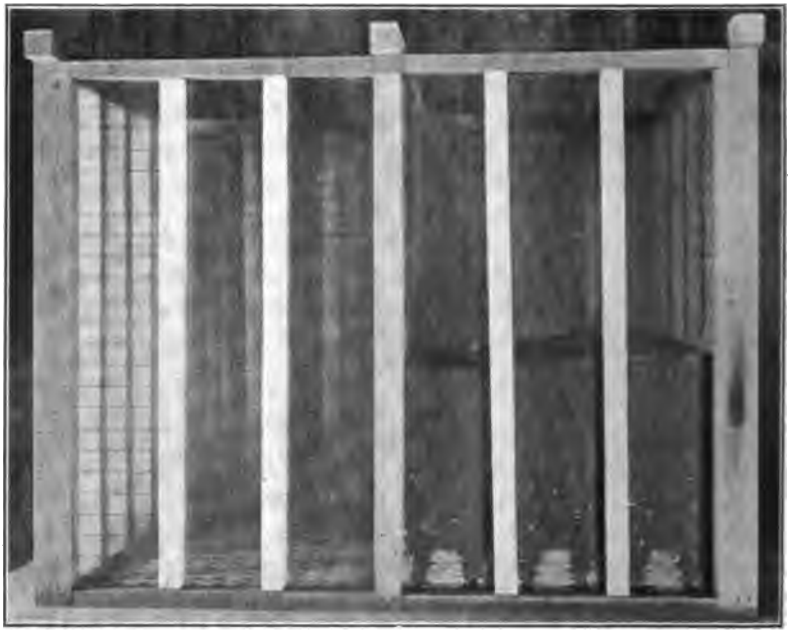


Fig. 1.—Experimental Box Filled with Crushed Ore and Capping; Drawn through Holes in the Bottom Representing Ore Chutes. Scale, 1 In. of Box =5 Ft. of Mine.

Conditions of Tests

In all, 16 tests or experiments were made in a period of 6 months beginning in the latter part of the year 1913. In all the tests the rock representing capping was nearly barren, containing only from 0.03 per cent. to 0.08 per cent. copper. The color of the capping was a dark red and that of the ore a light gray, thus giving a sharp and definite line between them, as clearly seen in Fig. 1.

In the first two experiments, weights and assays were not used. The ore in the box and products drawn from the chutes were measured by volume. They were run to get an idea, in a short time, of what could be done. In all the other tests the ore was weighed, sampled and assayed for copper, as were the chute products drawn during the test.

Tests Nos. 3 and 4 were made with fine and coarse ore mixed, the capping being a similar mixture of coarse and fine material. The conditions of test No. 3 represented 35 ft. of ore and 35 ft. of "capping" and those of test No. 4 represented 70 ft. of ore and 55 ft. of capping. Both tests were drawn at the closest chute interval, 6¼ ft. In both tests ore was drawn from one chute equivalent to about 10 tons, on the scale adopted, and then 10 tons from the next chute, and so on until 10 tons had been drawn from each chute. The drawing of all the chutes was then repeated, 10 tons at a time, until capping appeared in each chute. The total chute-drawings product was then weighed, sampled, assayed and called the clean-ore product. Drawing was then continued as before until each chute showed that it was running one-quarter capping. To determine this point, a sample mixture was kept in a test-tube and the chute product compared with it. This product was then weighed, sampled and assayed. Again the drawing continued until each chute showed about one-half capping. This point was also determined by comparing with a sample mixture and this product also weighed, sampled and assayed. The tests were concluded by drawing 10 tons from each chute in rotation, weighing, sampling and assaying the product. The assay value for the combined products of the tests was calculated, as the portions removed for assay could not be returned. Details of test No. 3 are given in Table 1.

TABLE 1.—*Details of Test No.* 3

Weights and assays—35 ft. of ore and 35 ft. of capping.
Ore and capping—coarse and fine mixed.
Largest boulder—1 ft. diameter.
Smallest particle—powder.
Drawn at 6¼-ft. centers—10 tons at one drawing.

Ore in box 127.50 lb.	1.96 per cent. copper
Capping	0.08 per cent. copper

Products	Pounds	Per Cent.	Per Cent. Cu
(1) Drawn clean to capping	81.00	63.52	1.96
(2) Drawn clean to ¼ capping	8.50	6.67	1.36
(3) Drawn clean to ½ capping	8.50	6.67	1.10
(4) Drawn 10 tons each chute	11.00	8.62	0.79
Totals and average	109.00	85.48	1.74

TABLE 1.—*Details of Test No.* 3 (*Continued*)

Products	Per Cent. Total	Grade, Per Cent. Copper	Product		Per Cent. Total Cu Recovered
			Per Cent. Ore	Per Cent. Capping	
Clean ore.......(1)	63.52	1.96	63.52	.....	63.52
Ore and capping (2)	6.67	1.36	4.54	2.13	4.54
Ore............(3)	6.67	1.10	3.62	3.05	3.62
	76.86	1.83	71.68	5.18	71.68

Estimate Applied to above Figures to Find Final Product of 1 Per Cent. Copper and Applied to Ores Below

Grade, Per Cent. Copper	Extraction, Per Cent. of Total	Grade of Product, Per Cent. Copper	Per Cent. of Total		Per Cent. of Product	
			Ore	Capping	Ore	Capping
2.00	79.500	1.842	72.96	6.54	91.77	8.23
1.90	78.340	1.763	72.44	5.90	92.47	7.53
1.80	77.180	1.681	71.84	5.34	93.08	6.92
1.70	75.490	1.605	71.07	4.42	94.14	5.86
1.60	73.490	1.529	70.06	3.43	95.33	4.67
1.50	71.490	1.449	68.92	2.57	96.41	3.59
1.40	69.290	1.367	67.54	1.75	97.48	2.52
1.30	67.290	1.283	66.35	0.94	98.61	1.39
1.20	65.290	1.195	65.02	0.27	99.58	0.42
Above Applied to Mining—25 Per Cent. by Development						
2.00	84.625	1.889	79.81	4.81	94.31	5.69
1:90	83.755	1.804	79.41	4.35	94.81	5.19
1.80	82.885	1.717	78.95	3.93	95.26	4.74
1.70	81.617	1.634	78.35	3.26	96.00	4.00
1.60	80.117	1.551	77.59	2.53	96.84	3.16
1.50	78.617	1.465	76.72	1.89	97.59	2.41
1.40	77.117	1.378	75.86	1.26	98.37	1.63
1.30	75.467	1.289	74.80	0.66	99.12	0.88
1.20	73.967	1.197	73.78	0.19	99.74	0.26

Besides estimating the extractions for different grades of ore, based on the experiments, the extractions for different grades of ore applied to mining were calculated, as shown in the lower part of Table 1, assuming 25 per cent. of the ore extracted by development and other work before the ore caves, when mining 35 ft. of ore, and 15 per cent. when mining 70 ft. of ore. For each grade, 1 per cent. copper was taken as the limit of commercial product. This point had to be interpolated from the product just above 1 per cent. and the product just below 1 per cent.

This way of estimating the results possible in mining a 1.80 per cent. copper ore, according to test No. 3 indicates a total extraction of 82.88 per cent. of 1.72 per cent. ore, 78.95 per cent. of which was original ore, the chute product being 95.26 per cent. ore and 4.74 per cent. capping.

In the next two tests, Nos. 5 and 6, the fines were screened out of the ore and capping, making both ore and capping represent on the scale of 1 in. = 5 ft., coarse chunks of 4 to 12 in. diameter. The chute interval and method of drawing were the same as in tests Nos. 3 and 4. Test No. 5 represented drawing 35 ft. of ore and test No. 6 represented 70 ft. of ore. Compared with tests Nos. 3 and 4, as shown in Table 2, both tests show higher extractions.

Tests Nos. 7 and 8 were duplicates of 5 and 6 except that the chute interval was 8⅞ ft. instead of 6¼ ft. These two tests show lower extractions than Nos. 5 and 6.

Tests Nos. 11 and 10 were the same as Nos. 5, 6, 7 and 8 except that the chute interval was 12½ ft. As will be noted in Table 2, still lower extractions were obtained with these two tests, showing conclusively that the greater the distance between chutes the less the ore extraction.

To determine the effect on extraction if no care is taken as to the amount drawn at one time, two tests, Nos. 12 and 13, were run. They were both made with conditions representing 35 ft. of ore and 35 ft. of capping, drawn at a chute interval of 8⅞ ft. In these two tests each chute was drawn direct to capping in rotation, then direct to one-fourth capping and then direct to one-half capping. Comparing these two tests with No. 7, with which all conditions were identical, except the method of drawing, it was found that in No. 7 a greater extraction of clean ore, before capping appeared, was obtained, but that in Nos. 12 and 13 a greater total extraction was obtained to the limit of 1 per cent. copper, as shown in Table 2.

Tests Nos. 14, 15 and 16 were special experiments made with conditions representing a chute interval of 8⅞ ft. and the column of ore 35 ft. In No. 14 the ore was coarse, 12 to 30 in. diameter, and the capping fine; a low extraction was obtained. Test No. 15 was the reverse of No. 14, the ore was fine and the capping coarse; the extraction was but very little higher than in No. 14. The last test, No. 16, was made with conditions representing both ore and capping fine. This apparently was the worst condition, the extraction being very low.

Conclusions Derived from Experiments

The conclusions derived from the experiments were as follows: (1) The less the distance between chute centers, the greater the extraction; (2) the higher the ore column, the greater the extraction; (3) the less the amount drawn at a time from each chute, uniformly, the greater the extraction of clean ore before the capping appeared, but not the greater the total

TABLE 2.—*Comparison of Extractions Based on 1.80 Per Cent. Copper Ore*

Test No.	Condition of Ore and Capping	Method of Drawing	Distance between Chute Centers, Feet	Depth of Ore, Feet	Extraction, Per Cent. Total	Grade of Product, Per Cent. Copper	Per Cent. Total		Per Cent. Product	
							Ore	Capping	Ore	Capping
3	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	77.180	1.681	71.84	5.34	93.08	6.92
4	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	93.950	1.705	88.76	5.19	94.48	5.52
5	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	86.400	1.717	82.30	4.10	95.26	4.74
6	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	95.030	1.765	93.13	1.90	98.00	2.00
7	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	35	78.570	1.717	74.85	3.72	95.26	4.74
8	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	70	89.550	1.764	87.71	1.84	97.94	2.06
10	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	70	85.110	1.741	82.24	2.87	96.63	3.37
11	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	35	67.390	1.707	63.81	3.58	94.69	5.31
12	Coarse, 4 to 12 in.	Direct to cap.—¼ cap.—½ cap. each chute.	8⅞	35	84.290	1.708	79.86	4.43	94.74	5.26
13	Coarse, 4 to 12 in.	Direct to cap.—followed by 10 tons each chute.	8⅞	35	83.980	1.703	79.33	4.65	94.46	5.54
14	Ore coarse, 12 to 30 in. Capping fine, 2 in. to powder.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	76.050	1.600	67.36	8.69	88.57	11.43
15	Ore fine, 2 in. to powder. Capping coarse, 12 to 18 in.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	80.040	1.564	69.24	10.80	86.51	13.49
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	53.940	1.635	48.85	5.09	90.57	9.43

Above Tests Applied to Mining 1.80 Per Cent. Ore—25 Per Cent. by Development for 35 Ft.; 15 Per Cent. for 70 Ft.

3	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	82.885	1.717	78.95	3.93	95.26	4.74
4	Coarse and fine mixed.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	94.857	1.720	90.52	4.33	95.43	4.57
5	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	35	89.800	1.740	86.72	3.08	96.57	3.43
6	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	6¼	70	95.775	1.770	94.13	1.65	98.28	1.72
7	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	35	83.927	1.742	81.15	2.78	96.69	3.31
8	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	8⅞	70	91.117	1.770	89.55	1.57	98.28	1.72
10	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	70	87.343	1.751	84.90	2.44	97.20	2.80
11	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	35	75.542	1.738	72.87	2.67	96.46	3.54
12	Coarse, 4 to 12 in.	Direct to cap.—¼ cap.—½ cap.—each chute.	8⅞	35	88.217	1.734	84.89	3.33	96.23	3.77
13	Coarse, 4 to 12 in.	Direct to cap followed by 10 tons each chute.	8⅞	35	87.985	1.731	84.52	3.47	96.06	3.94
14	Coarse ore, fine capping.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	82.037	1.661	75.53	6.51	92.06	7.94
15	Fine ore, coarse capping.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	85.030	1.633	76.92	8.11	90.46	9.54
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	65.455	1.698	61.64	3.82	94.17	5.83
		Average			85.38	1.723	81.63	3.75	95.16	4.39

extraction to a 1 per cent. copper final product; (4) the coarser the ore and capping without fines, the greater the extraction; (5) the higher the grade of ore, the greater the extraction, when a uniforn limit of grade is allowed at the end of the chute drawing. (6) Applied to mining, the results of the experiments indicate that the greater the amount extracted by development and undercutting, the greater will be the total extraction. This is on the assumption that when the ore caves it occupies the same volume as in place.

Some suggestions deduced from these experiments considered desirable to carry out in mining were: (1) Chute centers for drawing caved ore should be as near together as the ground will allow; (2) ore should be caved in as high columns as practicable; (3) when capping appears at the chutes, no more should be drawn unless the bottom slice is being mined.

While all of the tests described were made with a barren capping, in the actual mining of a sulphide orebody having mixed or oxidized ore over it in place of barren capping, the recovery of sulphide ore would be considerably increased over the experimental results, provided the mixed or oxidized ore could be treated at a profit.

In deciding what extraction could be expected by caving from 35 ft. to 70 ft. of ore in average caving ground, the results obtained in tests Nos. 10 and 11, as detailed in Table 2, were considered conservative, provided the chutes were closely watched and ore of below the commercial limit not drawn. If the ground under the caved ore, the location for raises and chutes, is hard and strong, results of tests Nos. 7 and 8 could be taken as obtainable. This opinion is based on the supposition that the capping will break up in about the same way as the ore and not into a fine powdery sand.

METHOD OF MINING

The Inspiration company's method of mining is one of the so-called Caving systems whereby the ore is caused to cave and crush itself, thus reducing to a minimum the blasting and handling. It is a modification of the method introduced by Felix McDonald at the Ohio Copper Co.'s mines in Utah, and was put into operation at the Inspiration under his supervision. The method consists, essentially, of undercutting the ore (taking out a horizontal slice), allowing the ore above to cave and crush, and drawing off the crushed ore through small inclined raises driven under the caved ore, into main inclined raises that lead down to the haulage-drift chutes.

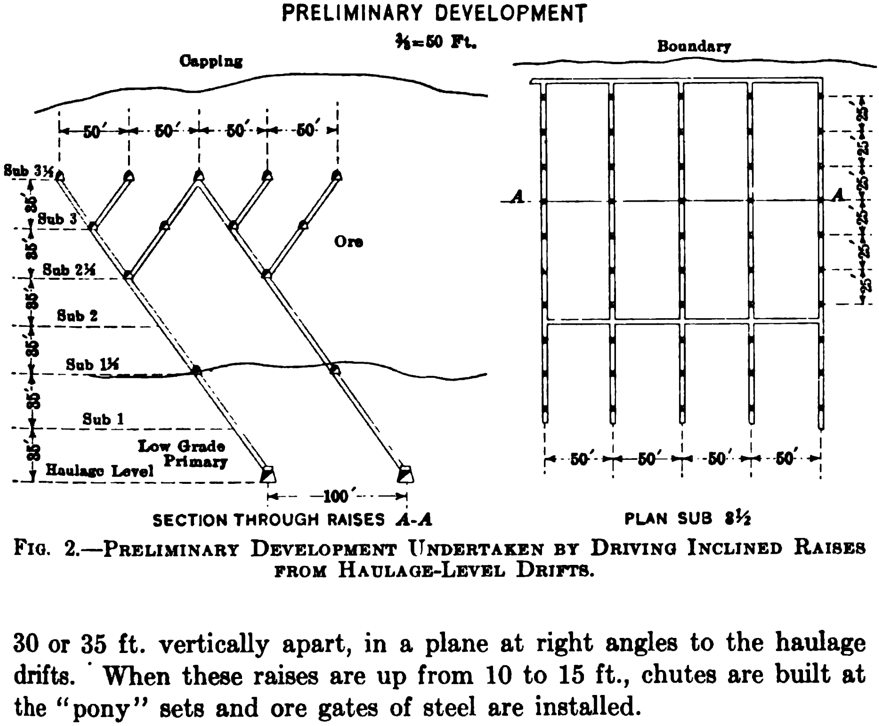
Preliminary Development Work

By referring to the accompanying illustrations showing the progressive steps of mine development, a good understanding of the method will be obtained.

After shafts have been sunk, the haulage drifts are driven under the ore of the section or sections to be mined at intervals of 100 ft. as shown in Fig. 2. These drifts are of large size, being 9 ft. wide at the base of the rails, 7½ ft. wide at the cap, and 7½ ft. high above the rail base. Where timbered, the above refer to inside timber dimensions.

After the haulage drifts are in, or during their driving, "pony" sets are put up every 25 ft. along the drifts. The "pony" set is about 5 ft. high, placed on top of the regular drift set and is the place from which the car loaders operate the chute gates discharging into the haulage cars.

Inclined raises are then started from the "pony" sets at an inclination of from 50° to 54°, depending on whether the sublevels are to be 30 or 35 ft. vertically apart, in a plane at right angles to the haulage drifts. When these raises are up from 10 to 15 ft., chutes are built at the "pony" sets and ore gates of steel are installed.



SECTION THROUGH RAISES A-A

PLAN SUB 3½

FIG. 2.—PRELIMINARY DEVELOPMENT UNDERTAKEN BY DRIVING INCLINED RAISES FROM HAULAGE-LEVEL DRIFTS.

The raises are then advanced until the first sublevel is reached. "Sub" drifts are then started, from the first raises to reach the required level, and are driven parallel to the haulage drifts. These drifts break into the other raises every 25 ft., as they advance, thus cutting down the cost of handling the drift "muck." The raises are then again advanced to the next sublevel where drifting is to start. The drifts on this level are then driven, meeting the raises as they advance. All of these development raises are about 4 ft. in diameter, and the subdrifts 6 by 7 ft.

This method of raising and sublevel drifting is continued until the height is reached where the first undercut is to be made, and the drifts are driven as before.

The first undercut sublevel will be located below the top of the ore, at about the height at which it is intended to cave. When the preliminary development is finished, the sublevel drifts on the first undercutting level are 50 ft. apart, as they are on the "sub" just below, as shown in Fig. 2. On all the "subs" below, the drifts are 100 ft. apart. All the drifts on each "sub" are also connected with inclined raises every 25 ft. Cross drifts, connecting all the drifts on each "sub," are also driven at intervals of about 150 ft.

The next work, before undercutting is commenced, is to drive on the level to be undercut other "sub" drifts, between and parallel to those already driven, as best shown by Fig. 3, making the drifts on this "sub" 25 ft. center to center. These drifts, as they advance, meet branch raises of the main inclined raises at 12½- or 25-ft. intervals as desired. These small branch raises, shown in Fig. 4, are called "finger raises," many of which are put up just before or ahead of the undercutting operations.

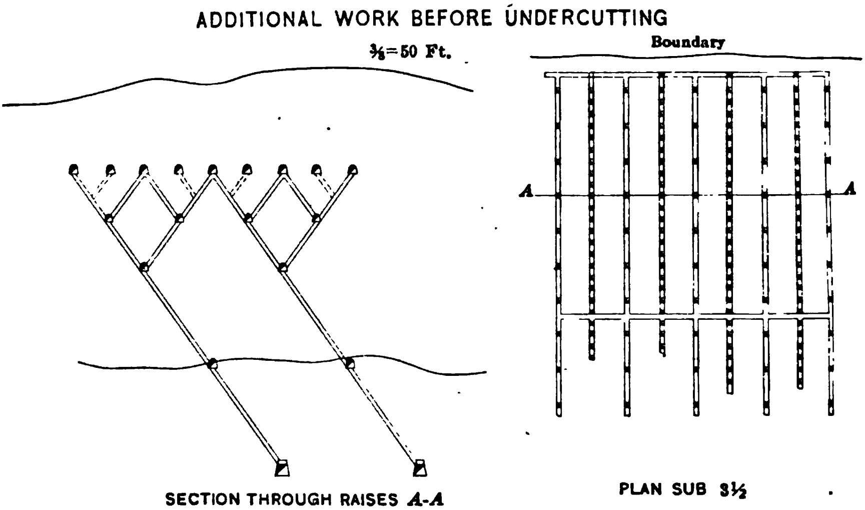

ADDITIONAL WORK BEFORE UNDERCUTTING
⅜=50 Ft.
Boundary
A
A
SECTION THROUGH RAISES A-A
PLAN SUB 3½

FIG. 3.—PRECEDING UNDERCUTTING, THE TOPMOST LEVEL IS DEVELOPED BY DRIFTS SPACED ON 25-FT. CENTERS AND CONNECTED WITH THE BRANCH, OR "FINGER RAISES."

Undercutting

Undercutting the ore is accomplished by starting at a cross drift, on the boundary of the section to be mined, and in retreating from that cross drift, as best shown in the plan of Fig. 5, drilling deep holes, at nearly right angles to the drifts, into the pillars between them, and blasting out the ground. Three holes are drilled in each side at different angles in the same vertical plane, with one hole in the back of the drifts, thus making seven holes to the round. For this work a water-hammer one-man drill with large steel is used. The holes are from 8 to 10 ft. in depth. Usually the rounds are blasted one at a time, the undercutting receding from the caved ground.

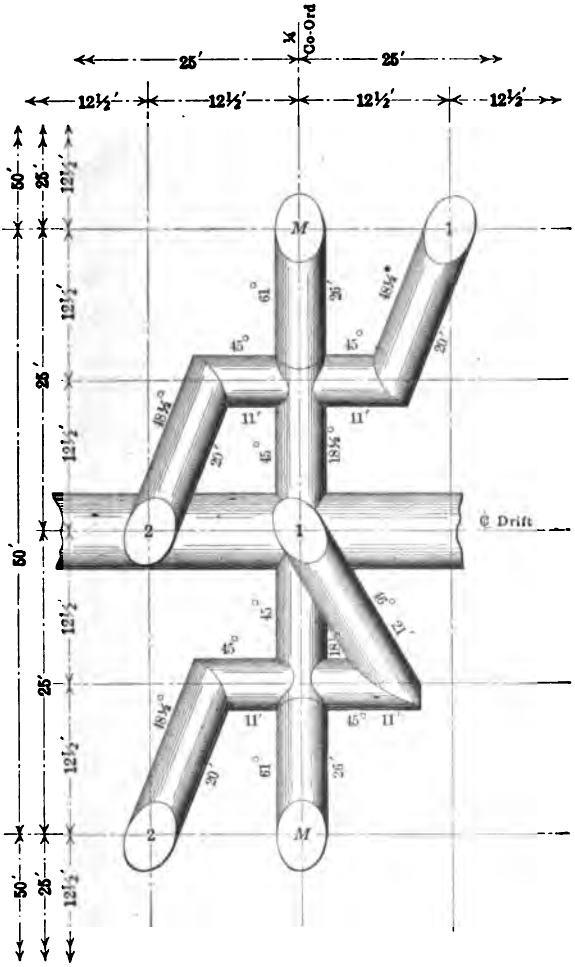


FIG. 4.—"FINGER RAISES", AN IDEAL TOP VIEW SHOWING THEIR CONNECTIONS WITH MAIN RAISES, *M*, AND SUBLEVEL DRIFT.

Ore from the undercut is drawn through the "finger raises," in each of which has been built an ordinary board chute to control the drawing. The "finger-raise" chutes are located about 4 or 5 ft. below the sublevel, so that they will not be blasted in shooting the undercutting holes.

While in some ground, the ore begins to cave as soon as it is undercut, in hard ground caving does not start until the undercutting has receded a considerable distance.

Ore Drawing

After the ore caves it is drawn off as desired. Large boulders that will not pass the raise chutes are blasted. In drawing off the ore, the chute "tappers" work in pairs. One "tapper" goes up a raise, from the "grizzly sub," which is the first "sub" below the undercutting level, opens a chute gate and draws the ore, while the other "tapper" works the ore through the grizzly on the "sub" below.

All grizzlies have about 1-ft. openings and are made of timber or steel rails. They are placed over all raises on the "grizzly sub" and are set at right angles to the drift. Pieces of ore too large to pass the grizzly are broken with an 8-lb. hammer by the "tapper" tending the grizzly.

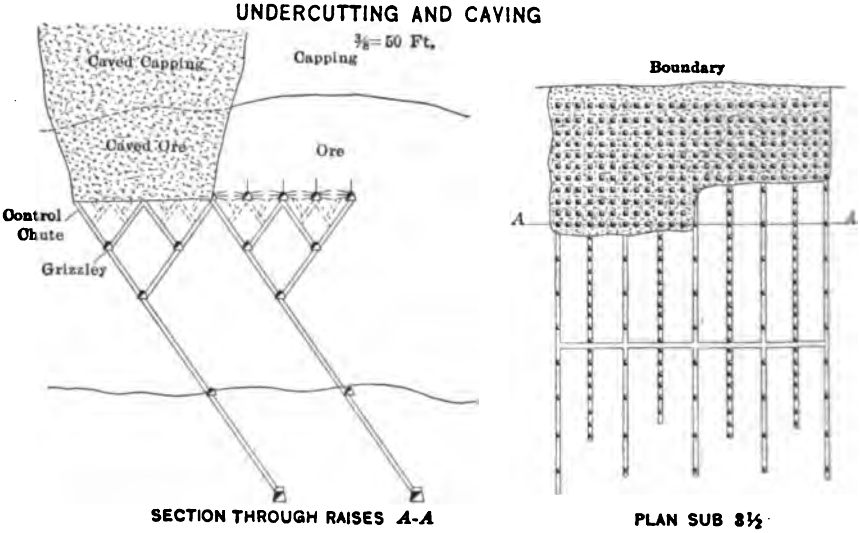


FIG. 5.—STARTING UNDERCUTTING BY BLASTING DOWN THE PILLARS BETWEEN SUBLEVEL DRIFTS.

After passing the grizzlies, the ore falls into the main raises and down to the haulage-drift chutes where it is loaded into 5-ton cars and hauled in trains, of 15 to 20 cars, to the shaft bins.

The second undercut can be located at one, or any number of sublevels below the first undercut, according to the height of ore which it is desired to cave. It could be located at the bottom of the ore if desired. In fact, the first undercut could be located at the bottom of the ore if it were known that the orebody to be mined could be caved throughout its total height.

Underground Haulage and Hoisting Arrangements

At the Inspiration mine the ore trains are hauled by compressed-air locomotives built by the H. K. Porter Co. of Pittsburgh. Arrived at the

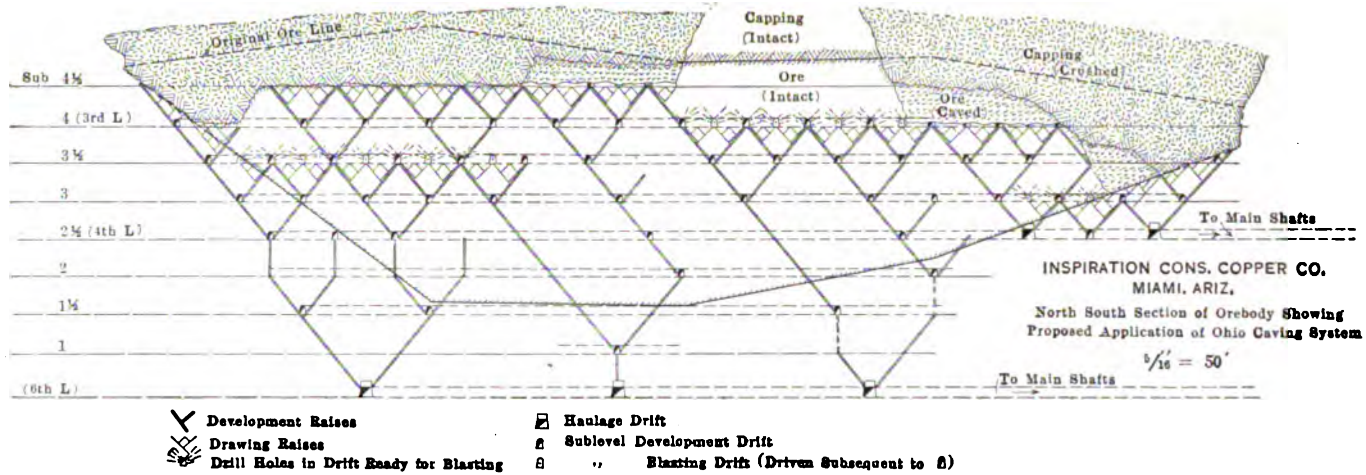

Original Ore Line
Capping
(Intact)
Ore
(Intact)
Capping
(Crushed)
Ore
Caved
Sub 4½
4 (3rd L)
3½
3
2½ (4th L)
2
1½
1
(6th L)
To Main Shafts
To Main Shafts
INSPIRATION CONS. COPPER CO.
MIAMI, ARIZ.
North South Section of Orebody Showing
Proposed Application of Ohio Caving System
5/16″ = 50′
Development Raises
Drawing Raises
Drill Holes in Drift Ready for Blasting
Haulage Drift
Sublevel Development Drift
″ Blasting Drift (Driven Subsequent to)

FIG. 6.—ARRANGEMENT OF MINE LEVELS AND RAISES UTILIZED IN INSPIRATION CAVING SYSTEM.

shaft, the cars are tippled into the shaft bins by barrel-shaped tipples, five cars at a time. From the shaft bins the ore is loaded automatically into 12½-ton skips and hoisted by electrically driven automatic hoists to the steel bins on the surface.

For hoisting ore, two main shafts 101 ft. apart are used. Both shafts have three compartments, of which two in each shaft are skipways. The third compartment in one shaft contains the ladderway, pipes and electric conduits, while in the other shaft the third compartment contains an Otis double-deck elevator for hoisting and lowering men. Both shafts, the underground bins, and the stations are all lined with reinforced concrete.

Only two levels are used for hauling ore, the 4th and 6th. The vertical distance between these two levels is 130 ft. A good idea of the arrangement of the mine is obtained from Fig. 6. On the 6th level the shafts are connected by two drifts, one for each shaft. In each drift there is a tipple over the bin at the shaft. The arrangement of the drifts, bins, tipples and stations is symmetrical with a center line between the two shafts so that the tipples are operated by one man from a central location.

On the 4th level one double-track drift passes between the two shafts and is connected to the shaft stations by a small cross drift. The ore is dumped by one tipple, on this level, into a small bin which is connected with the 6th level bins by a concrete-lined inclined raise. This arrangement makes necessary only one loading level for the skips.

The Inspiration Consolidated Copper Co. commenced underground development on a large scale toward the latter part of 1913 but did not start regular mining operations until August, 1915. The tonnage mined gradually increased as the concentrator was able to handle it until at present (June, 1916) an average tonnage of 16,700 tons is being mined daily.

The cost of mining is now 60 c. per ton, including 20 c. for development and all fixed charges.

The percentage of ore and copper extraction obtained by the Inspiration method cannot be determined positively until some section has been entirely mined from top to bottom of the ore. But based on results in six sections of the mine from the first undercut to the capping estimated to contain 1,886,450 tons, and which have been almost completely drawn to capping, the ore extraction is 102.44 per cent. and the recovery of copper 86.52 per cent. The second undercut is expected to increase the copper recovery.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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The Block Method of Top Slicing of the Miami Copper Co.

BY E. G. DEANE,[*] B. S., E. M., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

A METHOD of top slicing has been devised at the Miami Copper Co.'s mine at Miami, Ariz., which differs radically in some ways from the customary methods of top slicing.

The area of that section of the orebody in which top slicing is used is about 800 ft. square. The ore, while for the most part soft, is, nevertheless, considerably harder than the capping. The latter is siliceous, seldom containing any clay or other binding material, and breaks into fine particles so that it runs like sand if given the opportunity. Because of these facts, and because the ore is above the average grade of the mine ore, it has been mined by top slicing.

Haulage levels are opened up 150 ft. apart, vertically, with two sublevels between at 50-ft. intervals, to facilitate the building of ore chutes. These sublevels are used during slicing for distributing air in the ventilation system. On the haulage level the drifts are spaced on 50-ft. centers, and raises along these drifts are also spaced on 50-ft. centers, except the incline raises as hereafter noted. The raises are cribbed where necessary. Where the wear will be excessive, ¼-in. iron plates are spiked to the top of every third set of cribbing, for its protection.

When top slicing was first used, an attempt was made to carry a slicing face from fifty to several hundred feet long. Timber and other supplies were brought in through long drifts from an auxiliary shaft. Great difficulty was experienced in keeping these drifts open, the side pressure breaking the posts and the top weight breaking both caps and posts. Furthermore, the men could not work efficiently while these drifts were being repaired. The slicing faces advanced irregularly, due to varying conditions, and in many ways the results were not all that could be desired. It was then decided to divide the slicing area into blocks 200 ft. square and this was later changed to 250 ft. square.

At the center of each block a two-compartment raise is put up as a supply raise, the compartment being 2 ft. 6 in. and 4 ft. by 4 ft. 4 in., the smaller being used as a manway. Station sets of 12 by 12 timber with

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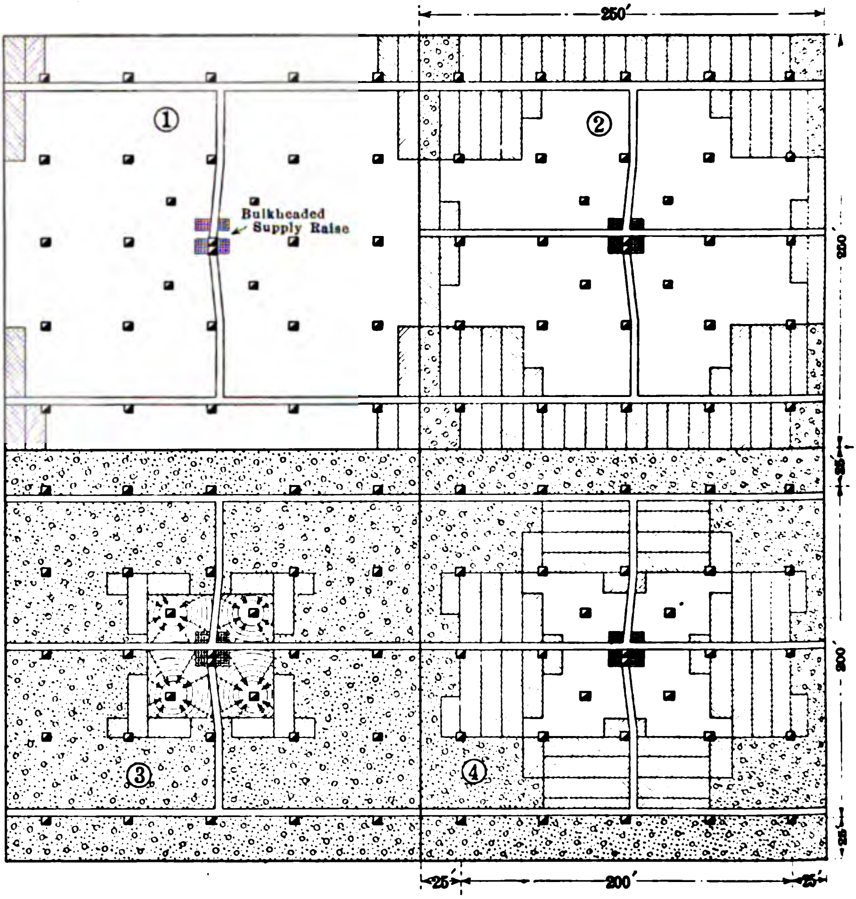
[*] Slicing Efficiency Engineer, Miami Copper Co.

9- or 10-ft. posts are put in and an Ingersoll stretcher bar air hoist is mounted above the larger raise compartment to use in hoisting the timber, steel, etc. Four bulkheads, built solidly of blocks of square timber, are put in as shown in Fig. 1. Two of these are 7 by 11 ft. in size, and the other two 7 by 7 ft.

Two drifts, usually untimbered at first, are run out 100 ft. on the long axis of the supply raise. At the end of each of these and at right angles to them, two drifts are run 125 ft. to the block limits. When these limits are reached, slices are started toward the corners of the block. These slices are timbered either with single sets consisting of two 8-ft. posts and a 12-ft. cap, or with a double set consisting of three 8-ft. posts and two 7-ft. caps, depending upon the ground. The ore is taken to the floor above.



Figs. 1 to 4.—Plan Showing Progressive Steps in Block Method of Top-Slicing.

As soon as the first slices have advanced a few feet, second and third slices are started, and also first slices toward the centers of the block limits. Every man who can work to advantage is put on and the ore mined with the greatest possible speed.

Where there is a sufficient mat of old timbers in the back to obviate the danger of the capping running into the ore there is no permanent floor laid, the planks used in shoveling and wheeling the ore being taken up later. Where there is no mat or it is not sufficient, a floor of 2-in. plank spiked to 2 by 10 sills is laid. Formerly 5 by 10 and 4 by 8 sills were used, but it was found that after being subjected to the pressure and heat of a completed stope, a 5 by 10 or 4 by 8 sill seemed to have practically no more strength than a 2 by 10.

As soon as the timbers in the slices show signs of taking weight, bulkheads are built of old timbers, obtained either from the mat in the back, or from repair work in other parts of the mine. As soon as possible, the posts are drilled and the slices shot down.

By the time slicing has started, four drifts have been run to the centers of the sides of the blocks as shown in Fig. 2 and slicing is also done from these. Working as intensively as possible, all ore except the four central pillars is quickly mined out as shown in the series of illustrations. As about the last of this ore is being taken, crosscuts are driven to incline raises, put up to about the center of the four central pillars, as shown in Fig. 3, and slicing continues, working from the outside of the remaining ore to the supply-raise bulkheads first put up as shown in Fig. 1, thus completing the stope. By this time, these bulkheads, which were 10 ft. high when put in, have squeezed to from 4 to 6 ft. in height. Upon completion, the stope is shot down and another may then be started below, though it is best to let the ground settle for a few weeks.

At first thought the criticism suggests itself that with such weight the mining method used intensifies this weight as the block approaches completion. But experience has shown that, as a rule, the maximum weight is taken by the timbers at about the time the outside pillars are completed, and as mining progresses a larger proportion of the weight is taken by the bulkheads in the outside slices. At no time does the weight on the remaining ore and the slices still necessarily open get beyond control. These central pillars constitute our cheapest ore, not only because of the pillar raises but because the ground has been fractured by the weight, and lifters are the only holes necessary to break the face.

All drilling is done with plugger machines, using a water spray attached to a 5-gal. can. No cars are used in the slices, all ore being shoveled directly into the chutes or wheeled in barrows. Round timber is used almost exclusively in the stopes, because of its superior strength.

Ten feet has been taken as the standard height of a slice. If a greater height is taken the ore sloughs off the top of the slicing face faster than it can be mucked out and the bottom shot, thus shortly caving the slice. It is possible that later, when a good mat has been formed, sublevel caving may be used, but so far, where tried, it has not been successful.

The ventilation of these slice blocks is very important because, without it, the heat coming out of the mat is excessive and prevents efficient work. It is accomplished by connecting one of the sublevels below the slicing floor with the discharge end of a 60,000-cu. ft. fan. Openings are maintained from the sublevel to raises through which it is desired to force air.

As a result of the change to the block method of slicing, and to using forced ventilation, the production per mucker-shift has been raised from 9 to 20 tons, the production per man-shift from 5 to 10 tons.

While the details of this method have been fully worked out and a large tonnage of ore has already been extracted by its use, the work to date has to a large extent been preparatory to systematic work for lower lifts. It is not possible to give representative costs, but the following is an estimate of what is expected:

Mining Costs per Ton

Preliminary development	$0.035	
Haulage development	0.025	
Other development (raises, etc.)	0.080	
Total development		$0.140
Stope Costs:		
Miners, at $3.75	0.080	
Muckers, at $3.75	0.160	
Drills	0.030	
Explosives	0.040	
Timbering, labor	0.070	
Timbering, supplies	0.130	
General—bosses, nippers, etc	0.040	
Total stoping		0.550
Haulage		0.055
Hoisting		0.04
Pumping		0.005
General underground		0.025
Ventilation		0.015
Engineering and sampling		0.016
Underground lighting		0.004
Mine surface		0.030
Total		$0.880

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Cost and Extraction in the Selection of a Mining Method

BY C. E. ARNOLD, B. S., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

In attacking the problems of mining and treating large disseminated copper orebodies such as those occurring in the Miami or the Ray district of Arizona, one of the vital questions to be decided is, "What is the value of the orebody per ton in place?" The importance of this question lies in the fact that its answer helps to reach a decision concerning the method of mining to be employed.

Let it be assumed for the moment that milling and smelting practice, especially the latter, is more or less standardized, and has a cost incapable of much variation, while the cost of mining has a considerable range owing to the large number of mining methods by which a mine may be exploited. For instance, a method may be adopted whereby all the available ore can be extracted perfectly clean (*i.e.*, unmixed with barren material) at a relatively high cost; or a method may be used involving the necessary dilution of a portion of the ore with waste, in order to recover that portion, thus giving a relatively poorer extraction but nevertheless a lower working cost than the former method. Naturally, between these two extremes there lie intermediate methods.

In order to arrive at a solution of a mining method problem that is of any value, it is necessary that accurate and complete estimates of mining, milling, and smelting performances and costs shall be established. By way of illustration is given a case of mining, by the system of undercutting and caving, a large orebody with an average content of 1.50 per cent. copper supposedly all in sulphide form, to which case has been applied those results in mining operations which one may fairly assume are to be expected when viewed in the light of G. R. Lehman's ore-drawing experiments, which form the subject of a paper to be presented at this meeting.

It is assumed that the total costs will be as shown in the following table:

Costs on 1 Ton of Clean Ore

Mining, including prepaid development	$0.600
Coarse crushing	0.024
Freight on ore	0.015
Concentrating	0.462
Freight on concentrate	0.002
Smelting, freight on copper, refining, marketing, etc	0.737
Amortization	0.120
Total	$1.960

To attain these costs, a daily production and treatment of 14,400 tons has been assumed. Other assumptions are as follows: 15 per cent. of orebody is to be extracted clean by development work and undercutting; 63.8 per cent. (75 per cent. of 85 per cent.) is to be extracted clean by caving, leaving 21.2 per cent. part of which is to be extracted by diluting with waste, and the remainder lost. Dilution and extraction are to be related as shown by the curve in Fig. 1. The copper tenor of the concentrate is to vary uniformly between 25.33 per cent. and 30.00 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. Mill extraction is to vary uniformly between 69 per cent. and 83 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. The selling price of copper is to average 14 c. per pound. The life of property will be 12 years. Smelting is to be performed under contract with an outside company.

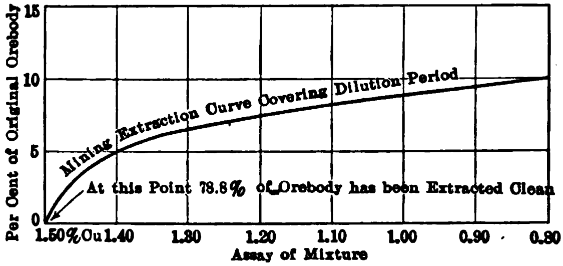

FIG. 1.—SHOWING EFFECT OF DILUTING THE LAST DRAWN ORE WITH CAPPING UPON COPPER ASSAY OF THE MIXTURE.

In the above set of assumed costs the charge of $0.12 per ton of ore for amortization represents the outlay necessary to redeem in 12 years all expenditures for plant (not including smelter), lands, railroad, water supply, etc., other than the expenditures required in preparing the mine for the assumed daily production. This latter expense is covered by a charge of $0.20 per ton, which is included in the estimate of $0.60 per ton for mining. The amortization fund is considered as an annuity invested at 4 per cent.

It should be borne in mind that as long as the ore is undiluted with waste, the above total cost of $1.96 is all that is applied to the ton in place, and that the instant dilution starts the cost per ton in place increases because the added waste, until it disappears in the mill tailing, goes through all the operations to which the ore is subjected. Furthermore, when the ore is diluted, the mill extraction, and consequently the smelter return, decreases, likewise the copper tenor of the concentrate decreases. These disadvantages, however, are to a certain extent offset by the fact that on dilution, the increasing concentration ratio and the decreasing copper content of the concentrate lower the smelting cost on the ton of ore in place.

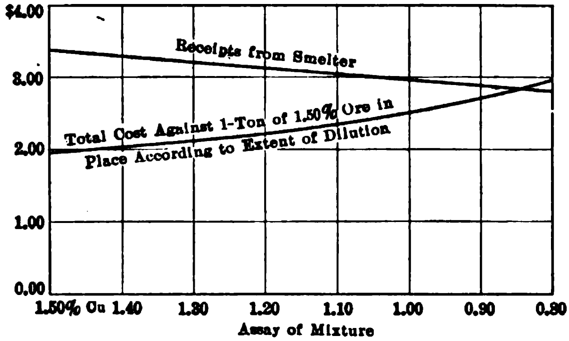

FIG. 2.—SHOWING DECREASING SMELTER RECEIPTS AND RISING COSTS WITH INCREASED DILUTION.

Taking into account all these variables, the curves shown in Fig. 2 were plotted, and then by using these in conjunction with the curve of Fig. 1, the curve of Fig. 3 was established, showing the extent to which dilution could be carried to secure the maximum profit from that ore remaining in place after the extraction of all available clean ore.

The conclusions resulting from this set of calculations are as follows:

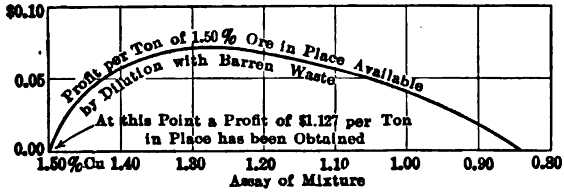

FIG. 3.—SHOWING MAXIMUM POSSIBLE PROFIT WITH ORE DILUTION INCREASED UNTIL ORE ASSAY IS REDUCED TO 1.28 PER CENT. OF COPPER.

1. To obtain a maximum profit the average assay of the dilution product will be 1.28 per cent. copper, the end point being 0.845 per cent., as shown in Fig. 3, at which point receipts from sales of copper just balance total costs.

2. The total ore recovered will be 85.5 per cent., as shown in Fig. 1, of which 78.8 per cent. will be extracted clean and 6.7 per cent. will be extracted in conjunction with 1.15 per cent. of waste.

3. From 100 tons of ore in place there should be recovered:

78.8 tons clean, at a profit of $1.430	$112.68
6.7 tons diluted, at a profit of 1.063	7.12
85.5 tons at a total profit of	$119.80

Expressed otherwise, the above might be stated thus:

78.8 tons clean ore at a profit of $1.430	$112.68
6.7 tons clean ore at a profit of 1.273	8.53
Gross profit	$121.21
1.152 tons waste at a loss of 1.221	1.41
Net profit	$119.80

This is equivalent to $1.20 per ton in place.

To duplicate this performance by the employment of a mining system that would give, say a 95 per cent. extraction of clean ore (this representing the end of mining operations), a mining cost increased from $0.60 up to $0.77 per ton would have to be borne as indicated below:

	Per Ton
Receipts from smelter	$3.39
Costs other than mining	1.36
Difference	2.03
Cost of mining	0.77
Difference	1.26
Mining extraction, per cent	95
Profit per ton in place	$1.20

Further, it may be stated that on some particular ore, of higher copper content than 1.50 per cent., it would be equally profitable to apply either a low-cost, low-extraction mining method, or a high-cost, high-extraction method; and with further increase in the copper content of the ore the high-cost, high-extraction method would yield the greatest profit per ton in place.

By the application of similar reasoning to various mining problems such as that outlined above, it will be found that the resulting figures indicate plainly the mining method that will yield the greatest ultimate profit. Should the calculations show that there is little difference in the results of several methods, the logical decision would be to use that method in which the element of doubt is smallest regarding the realization of the estimated costs.

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Shaft Sinking through Soft Material

BY EDWARD A. SAYRE, E. M.,* DES MOINES, IOWA

(Arizona Meeting, September, 1916)

In shaft sinking for coal mines, the cost item greatly influences the method adopted. This holds true especially when soft material must be traversed. The average life of a coal mine is short. This is due either to the limited area of the coal basin or to the great expense of maintenance and haulage underground when the entries are extended a considerable distance from the hoisting shaft. Therefore, the engineer in opening a coal mine is confronted with the fact that the cost must be kept within certain bounds else the venture will prove unprofitable. For this reason he may be prohibited from adopting certain safe, but expensive, methods that are used in sinking caissons for foundations, driving transportation tunnels, or other work of permanent construction.

A common method of sinking through difficult ground employs the aid of a steel shoe pushed ahead of the shaft timbers. Another is the drop-shaft method. These two methods were used in sinking the main shaft and the air shaft of the Eagle No. 3 mine, at Des Moines, Ia., and the following data show the relative success and cost of the two methods of sinking under the same conditions.

A drill log of the material to be penetrated was as follows (see Plate 1, Fig. 1):

	Thickness, Ft.	Thickness, In.	Total Depth, Ft.	Total Depth, In.
Drift	38		38	
Sand	2		40	
Drift	2	6	42	6
Jointed clay	5	6	48	
Drift, firm	10	6	58	6
Clay, with sand streaks	3		61	6
Sand	12		73	6
Shale, with two thin beds of coal and thin strata of rock	86	6	160	
Rock	5		165	
Coal	3	9	168	9

Main Shaft Sunk by Steel-Shoe Method

This record shows the first 73 ft. to be of drift material, the lower 12 or 15 ft. of which was sand. The remaining 92 ft. to the coal was shale and rock, and offered no trouble in sinking.

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[*] General Manager, Eagle Coal & Mining Co.

The general equipment, purchased from a company that had twice attempted to sink on this property and failed, consisted of one 80-hp. boiler, a hoisting engine, one No. 5 horizontal and one No. 6 vertical

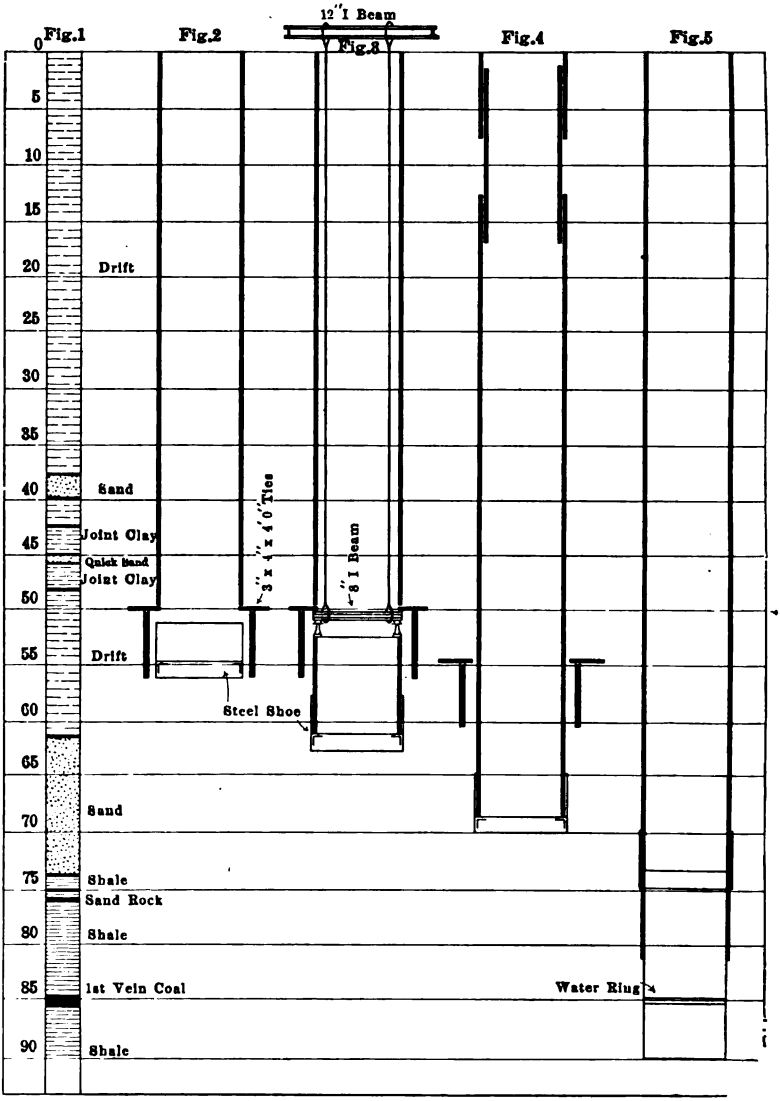

PLATE 1.

Cameron pump, a dump car, hoisting buckets, etc. The men employed were a day and a night engineer, one timberman, and three sinking crews, each consisting of three sinkers and one top man. Work was continuous 24 hr. a day until the sand was reached, when it was cut to two shifts, the remaining time being used in repacking pumps, etc.

The excavation was made 8 by 14 ft. The curbing for the first 50 ft. consisted of 2 by 4 and 2 by 6 timber laid flatwise, alternately with a quarter shaft bratticed off with 3 by 12 material, and the hoisting compartments divided by 4 by 6 buntons.

No trouble was experienced in sinking until the bed of sand at the 38-ft. level was encountered. This sand carried a small amount of water and, as a shaft protection, a concrete ring was put in behind the curb. The next trouble was experienced from the 42- to 48-ft. level. It had been the intention to assemble the shoe at this level, but what appeared to be a firm green clay proved to be jointed clay with a seam of quicksand through it and the exposed wall would not stand. Temporary 2 by 12 curbing was put in place and then regular curbing carried on down.

At the 50-ft. level, in order to safeguard the assembling of the steel shoe, 125 wooden mine ties, 3 by 4 in by 4 ft. long, were sledged into the clay just below the curb as shown in Plate 2. Steel angles, 6 by 4 in., were fastened with lag screws to the under side of the ties. An excavation 6 ft. deep was then made, undercutting the ties 16 in., and a false curb of 2 by 6 boards built. The shoe was then assembled at this point.

The shoe consisted of four sheets of ⅜-in. steel, two 8 ft. long by 5 ft. high and two 14 ft. long by 5 ft. high, with 3-in. angles at the corners, and a 6-in. pressure angle placed 18 in. from the bottom of the shoe. The curbing was then built inside the shoe to a height permitting the jack screws to be placed against the curb timbers of the shaft, as shown in Plate 2.

The clay was then excavated, the shoe lowered into the sand, and pumping started. The sand carried about 60 gal. of water per minute, which would rise about 18 ft. above the sand when pumping was stopped.

In order to avoid the formation of cavities behind the curb, as little sand as possible was excavated. To prevent an inrush of sand under the shoe, at least 1½ to 2 ft. of sand had to be left within the shoe. The method of sinking was to agitate the sand at the bottom of the shoe and force the shoe through it by means of the jack screws bearing against the shoe and the curb of the shaft. Two methods were used to agitate the sand. First, while the men could reach the bottom of the shoe, they stirred the sand with spades. By this method the shoe was lowered about 18 in. in the sand. Later the pumping system shown in Plate 2 was used.

In this second process the discharge of the No. 6 pump could, when desired, be sent through five ¾-in. pipes, as indicated in the drawing, and these five jets of water could be played upon the sand at the bottom of the shoe. It was found that these jets would agitate the sand sufficiently to permit the jack screws to push the shoe down, except when the sand was at too high a level inside the shoe, in which case sand would have to be excavated before the jet process could be resumed. In using this process the men would stir the sand with the jets for about 10 min., and then tighten the jack screws. When sufficient space was obtained between the timbers in the shoe and the curbing above, the jack screws of one side were removed, and sets of timber put in place on top of the timbers in the shoe, and then the other jack screws were moved. An advance of 18 in. per day was exceptional by this method. More often it was less than a foot.

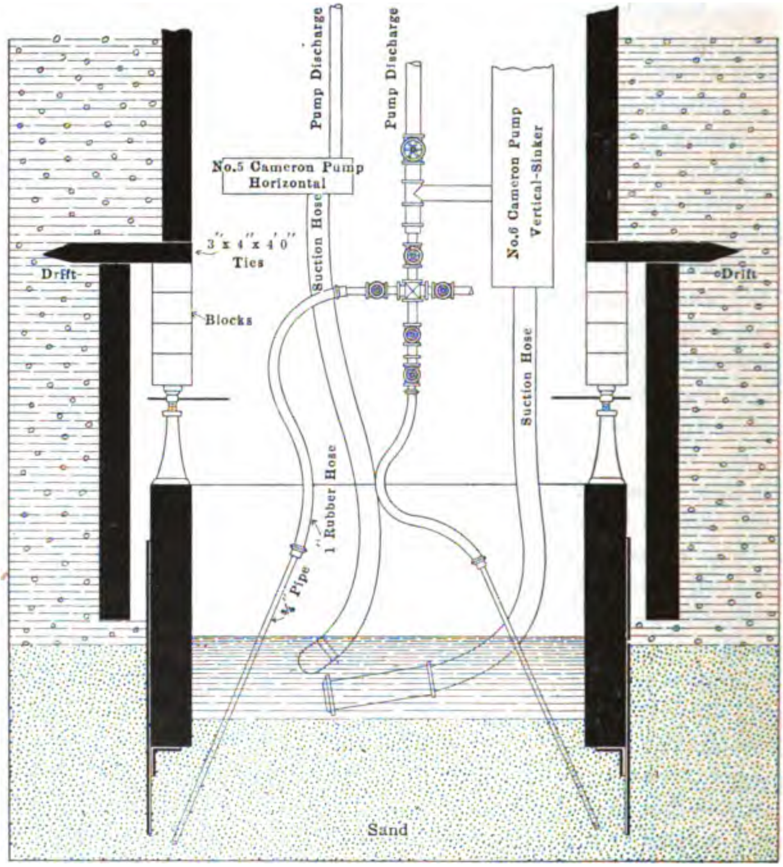

PLATE 2.—DETAILED SKETCH OF SINKING SHOE, JACK SCREWS AND PUMPS.

Necessarily some sand had to be excavated at times. This had a tendency to cave the dirt around the shaft, which in turn caused an excessive down pressure and broke the curbing apart a number of times.

An attempt was made to overcome this trouble by supporting the curb with I-beams and cables from the surface as shown in Plate 1, Fig. 3. Ten 12-in. I-beams were supported on cribs at the surface, and ten 8-in. I-beams were swung under the 4 by 6 angle below the ties, these beams being connected by twenty ⅞-in. steel cables. When the sinking was resumed, the curbing continued to break, the I-beams bent, and two cables were broken. Since it appeared impossible to hold the curb, it was decided to timber the shaft solid from the 8-ft. level (where the most uniform break occurred) to the bottom of the shoe, and then drop this portion of the shaft through the remaining 5 ft. of sand.

To do this, the ties were driven back into the wall and solid timbering put in between the shoe and the upper curbing of the shaft. The entire shaft curbing from the 8-ft. level down was then tied together with 2 by 6-in. stringers. At the 8-ft. level, 2 by 4-in. by 16-ft. planks were spiked to the lower curb, the upper ends projecting above the break as shown in Plate 1, Fig. 4, preventing the loose material from falling down the shaft. The jet system, with occasional excavations of sand, was resumed, and the shoe, with 50 ft. of curbing, was lowered through the sand.

In landing the shaft on the solid, seven boulders from 1 to 2 ft. in diameter were encountered. Six of these were under the cutting edge of the shoe, and were removed only after being broken up by means of a long chisel and sledge.

While lowering the shaft through the sand, the upper timbers of the of shaft buckled 18 in. out of line. This necessitated retimbering of the shaft from the top of the sand to the surface, an expensive undertaking because the old timbers had to be cut out and replaced in sections.

On completion of the retimbering, sinking through the shale was commenced. Three shifts of four sinkers each were used with an average daily advance of 5 ft. The only problem involved in sinking through the shale was the elimination of the water, a considerable proportion of which was choked off when the solid was reached. This water was taken care of by placing a water ring at the 85-ft. level with a pump located at that point to elevate the water to the surface as shown in Plate 1, Fig. 5.

Air Shaft Sunk by Drop-Shaft Method

The air shaft was located 350 ft. from the main shaft, and its sinking conditions were similar except that the surface at this point was 10 ft. lower than at the main shaft, making the actual distance to be traversed to the solid 63 ft. 6 in.

The equipment was the same as that used at the main shaft. The air shaft followed a drill hole tapped by an entry from the main shaft, so that most of the water was drained through this drill hole and then pumped to the surface. Two shifts of three sinkers, one top man, and an engineer were employed.

The size of the excavation was the same as before, 8 by 14 ft. The steel shoe was similar to the one in the main shaft, except that it was 10 ft. high instead of 5 ft. The timbering for the first 30 ft. above the shoe consisted of 4 by 6 laid flat, tied together by lag screws ⅝ in. diameter by 10 in. long, spaced 2 ft. apart. The shaft was divided into three equal compartments by 4 by 6 buntons as shown in Plate 3. The middle compartment was left free and was used for hoisting purposes. The end compartments were braced by 4 by 6 extending from the ends of the shaft to the sides. Four steel straps ⅜ in. by 2 in. tied the shaft timbers together from top to bottom.

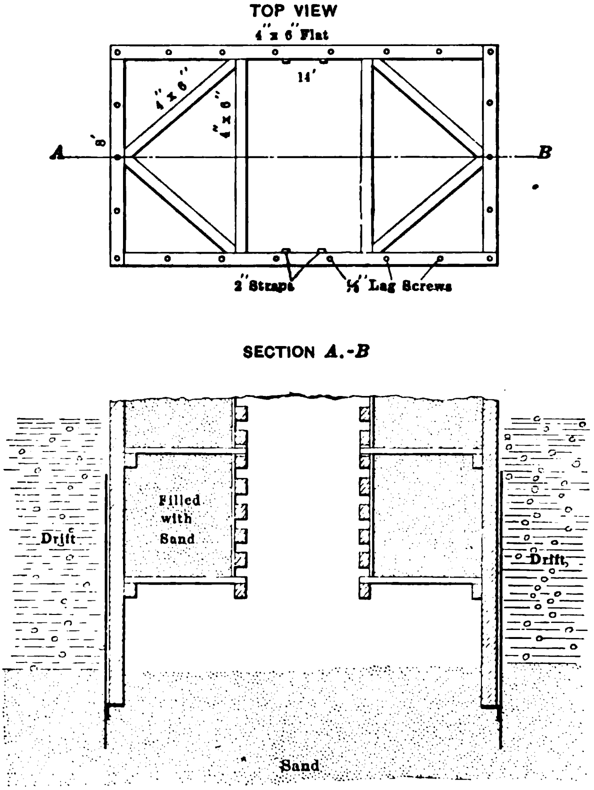

PLATE 3.—METHOD EMPLOYED IN SINKING AIR SHAFT.

An excavation 10 ft. deep was first made, the shoe assembled and lined with timbers. The sinking was continued until the shoe was hung up by the friction on the sides. Then a platform was built every 5 ft. in the end compartments. These 5-ft. chambers were filled with sand to give additional weight, and the sinking continued. The ground surrounding the shoe gradually broke in an oval shape. At one place it was necessary to fire a small charge of powder in order to loosen the ground sufficiently at one end of the shaft.

As the shoe was sunk, timbers were added at the top of the curbing. This method of building the curb at the top is decidedly better than that of adding timbers at the bottom since the timbers are placed much more accurately and expeditiously.

One difficulty experienced in the drop-shaft method was to keep the bottom of the shoe level. When one side of the shoe got lower than the other it kicked the opposite side outward. To right it, the lower side was blocked until the higher side caught up.

The progress through the drift material, until the sand was reached, was slow, much slower than at the main shaft. The drop-shaft went much faster, however, after reaching the sand. In fact, the difficulty at that time was to keep the bottom of the shaft from moving faster than the top. When within 10 in. of the bottom of the sand, the shaft broke apart 20 ft. up from the shoe. This was due to the fact that the movement of the shoe was faster than that of the top of the shaft, and to the insufficient strength of the straps connecting the top and bottom of the curb. At this point (20 ft. above the shoe) the curb separated from 6 to 8 in., and the upper part of the shaft kicked over 9 in. north and east of the lower part. A temporary platform of 8 by 8 timber was put in the end compartments of the shaft at this place, and time given for the upper part of the shaft to settle down before starting the excavation again. The sinking was then continued and the shoe landed on the solid without further difficulty, aside from hitting two small boulders at the bottom of the sand.

As the excavation was larger than necessary for an air shaft, it was decided to cement the shaft for a distance of 28 ft. from the bottom of the shoe, in order to shut off the water. A wall of cement 4 to 8 in. thick was accordingly then constructed.

After the cement was given time to set thoroughly, the excavation was again started in the shale and continued without difficulty to the coal. Sinking through the shale in the air shaft cost slightly more than in the main shaft because work in the mine prevented as careful supervision being given the sinkers in the air shaft.

One difficulty encountered in drop-shaft sinking was in keeping the position of the shaft vertical. At one time this shaft was 2 ft. out of plumb. By regulating the movement at the bottom of the shoe, the shaft partly righted itself, until at the finish, in a total depth of 63 ft. 6 in. to the shale, the bottom of the shaft was 16 in. to the south and 10 in. to the east of the top. Part of this variation was remedied in the cementing of the shaft.

A much larger amount of sand was removed in sinking the air shaft by the drop-shaft method than in sinking the main shaft. This could be done without danger of a cavity forming, because the surface dirt followed the air shaft down as it descended. When the sinking through the sand was completed, the surface directly surrounding the air shaft had caved to a depth of 15 to 16 ft. and for a distance of 20 ft. in all directions from the shaft. In fact, all the shale that was removed through the remaining 92 ft. to the coal did not fill this space at the surface.

A comparison of the costs of the two shafts is as follows:

	Main Shaft		Air Shaft	
Labor:				
Through drift material	$916.80		$789.08	
Through sand	1,941.80		541.62	
Through shale	1,065.68		1,213.30	
	$3,924.28		$2,544.00	
Superintendence	600.00		435.50	
Retimbering	1,343.23			
Cementing	....		207.68	
Total labor cost		$5,867.51		$3,187.18
Materials:				
Curbing	$1,878.62		$1,195.17	
Supplies	900.14		642.74	
Power, light, water, insurance, etc	1,248.50		649.71	
Total curbing, etc., cost		$4,027.26		$2,487.62
Total costs of shafts		$9,894.77		$5,674.80

Conclusions

In this particular work there was no question about the superiority of the drop-shaft method of sinking. It made a net saving of $4,300 in the total cost of the air shaft compared with the main shaft. A saving of $2,700 was effected in the labor cost, while in the cost of materials, power, etc., the saving was $1,600. A saving in time also resulted, 30 days being required to traverse the sand with the main shaft, while the air shaft was dropped through it in 17 days.

From the results obtained in these two shafts, and from the experience of others in the western interior coal field, we believe that the drop-shaft method of sinking is the safest, most economical, and most successful that can be adopted for sinking through soft material that lies within 100 ft. of the surface. At greater depths a variation of the method can be used by first sinking a larger shaft close to the soft material, and then telescoping a drop-shaft within it.

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Power Plant of the Burro Mountain Copper Co.

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(Arizona Meeting, September, 1916)

The power plant of the Burro Mountain Copper Co. is located near Tyrone, N. M., at 5,950 ft. elevation. It is interesting because it uses the largest stationary Diesel engines in the United States.

The general layout of the plant is shown in Figs. 1 and 2. The building is of steel-frame construction with hollow-tile walls and corrugated-iron roof; the floor is concrete and all windows have steel frames, making a fireproof building.

The building is made to accommodate one more electric generator unit which is being installed; also two 4,000-cu. ft. air compressors direct-connected to three-cylinder engines with the same size cylinders as the electric units.

The power plant consists of two 815-kva., 60-cycle, three-phase, 6,600-volt, 180-r.p.m. generators, direct-connected to five-cylinder, two-cycle Diesel engines; the necessary circulating pumps for cooling water, cooling towers and oil storage tanks.

A portion of the power is used in the same building as the generating units to run one of two 200-kw. rotary converters delivering 260-volt direct current, and a 2,500-cu. ft. air compressor, delivering air at 95 lb. pressure, direct-connected to a 400-hp., 6,600-volt, three-phase synchronous motor.

The remainder of the power is used to run a 1,500-ton concentrating mill 3½ miles from the power house, various hoists, pumps and motors about the mines and the lighting and water-works of the town of Tyrone.

The direct current is used for electric traction in the mine, also to run hoists and motors that were installed before the new power plant was built.

Due to the large hoists that take starting peaks of 250 kw. and the 10-ton electric locomotive that takes at times 150 kw., the load is extremely variable, as shown in Fig. 3.

The two five-cylinder vertical Diesel engines, which are rated at 1,250 b.hp. at sea level, have cylinders of 525-mm. diameter (20.6 in.) and

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* Consulting Engineer, Phelps, Dodge & Co.

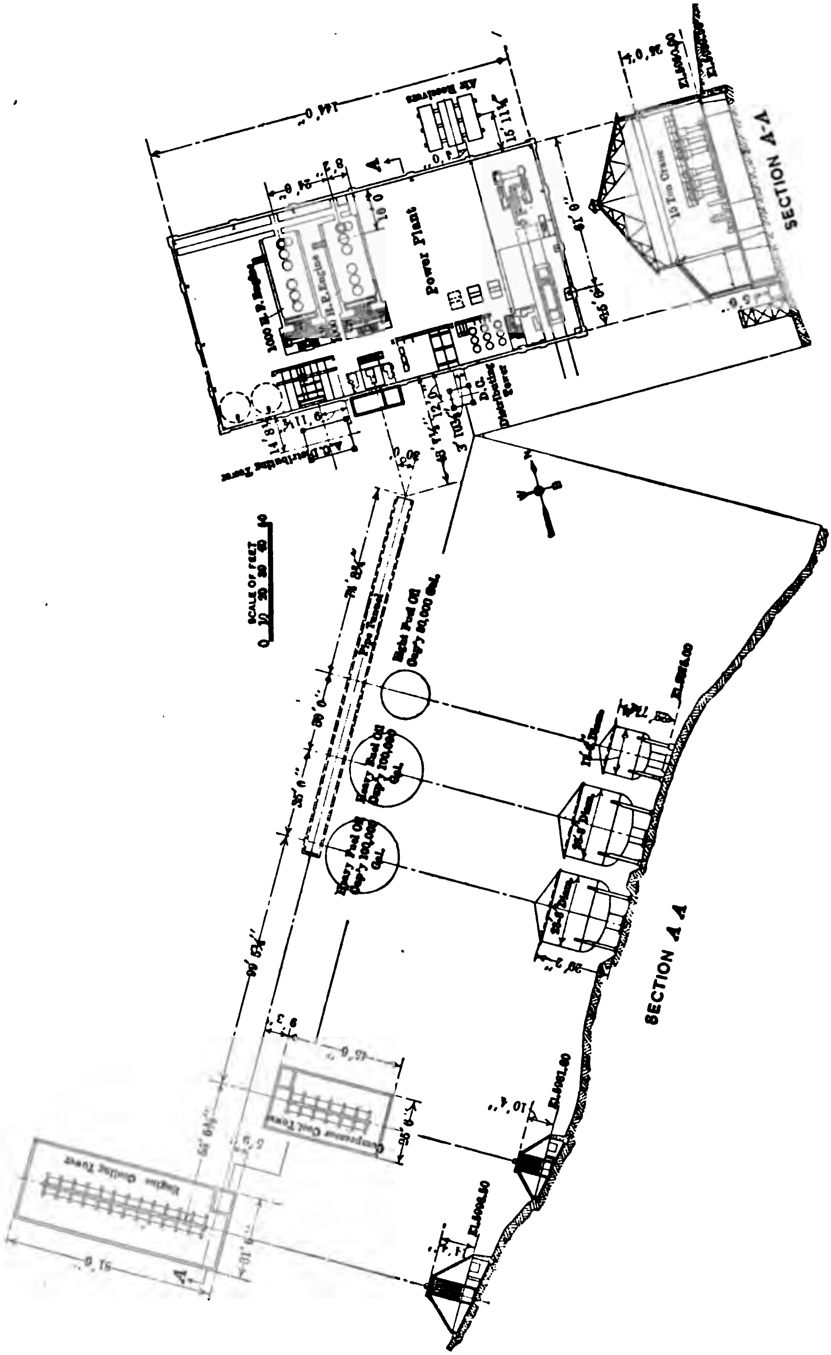

FIG. 1.—GENERAL ARRANGEMENT OF POWER PLANT EQUIPMENT, BURRO MT. COPPER CO., TYRONE, N. M.

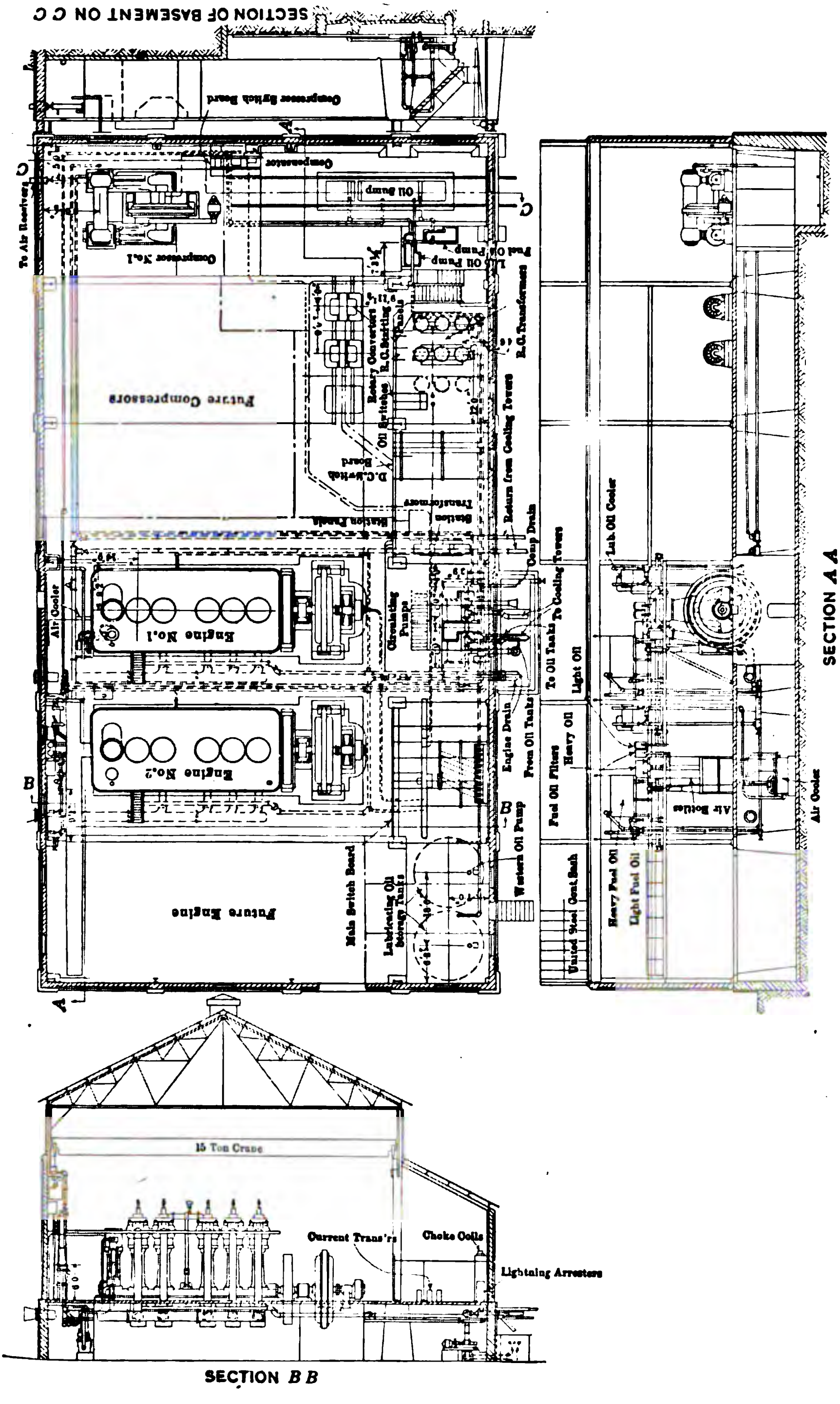

FIG. 2.—GENERAL ARRANGEMENT OF POWER PLANT OF BURRO MT. COPPER CO.

660-mm. (26 in.) stroke; each engine has a scavenging cylinder of 1,050-mm. (41.25 in.) diameter and 600-mm. (23.6 in.) stroke; also a three-stage, four-cylinder high-pressure vertical compressor, both directly connected to the engine. This compressor delivers the air necessary for fuel injection and for starting the engine. The scavenging pump, which is larger than usual, delivers the air to blow off the products of combustion and fill the cylinders with fresh air at the beginning of the stroke This pump was increased in size to be able to fill the cylinders with air at 2½ lb. gage pressure at the beginning of the stroke. This gives nearly the same initial absolute pressure and allows the engine to generate nearly the same indicated horsepower as it would at sea level. The work done in the scavenging pump is, however, increased and the horsepower available is approximately 95 per cent. of sea-level output.

The fuel consumption per horsepower is increased over sea-level conditions because of the extra work of the scavenging pump and the resulting lower mechanical efficiency of the engine when operating at the elevation of this plant.

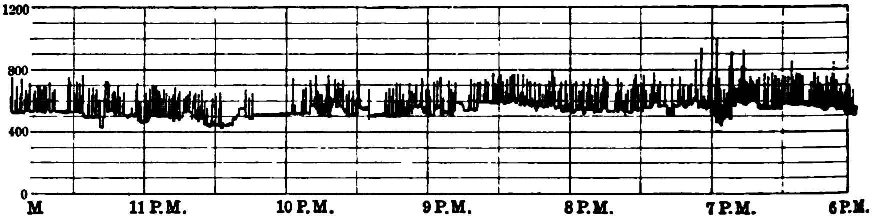

FIG. 3.—SECTION OF CHART SHOWING VARIATION IN POWER LOAD.

The plant has been in operation for 17 months, but as the mill was not ready until the middle of April, 1916, up to that time the load was carried on one engine.

The fuel used is California asphaltum base oil 14° to 18° Bé. gravity, averaging 16° Bé. and 18,360 B.t.u., costing $1.85 to $1.98 per barrel. The number of heat units in the oil was not determined at Tyrone, but at another plant which gets the oil from the same shipping point and under the same contract.

The pistons, cylinders, heads and exhaust pipes are water-jacketed; the jacket water of the exhaust pipes, which can be easily varied in temperature without danger, is used to heat the fuel oil to 120° F. as the heavy fuel oil cannot be used if cold. The engines are started with a lighter fuel oil of about 25° Bé. and run on this oil until the heavy oil is heated to the required temperature; when shutting off an engine, it is run for a few minutes on light oil to fill up the oil piping with light oil.

The curve in Fig. 4 shows the oil consumption per kilowatt-hour delivered at the switchboard when engines were tested new; it is the average of several tests. It is interesting to compare the test figures with the actual consumption as shown on power reports of the company. Although individual months show variations from the curve, the yearly average for 1915 checks very closely with it, showing that the efficiency of the engines is well maintained and that the variable load has very little detrimental effect on the fuel efficiency.

By the courtesy of Phelps, Dodge & Co., owners of the Burro Mountain Copper Co., I am allowed to publish in the accompanying table the operating costs from Jan. 1, 1915, to June 1, 1916. These costs are as they stand on the books and are higher than is to be expected. They cover the running cost of the complete plant, exclusive of the air compressor. These costs do not include taxes, overhead charges or depreciation. The cost per kilowatt-hour available is based on total power generated less that used for auxiliaries in power plant, such as lighting, circulating pumps for oil and water, etc. To simplify bookkeeping, the compressor is not charged with the cost of circulating its water or with its proportion of the lighting.

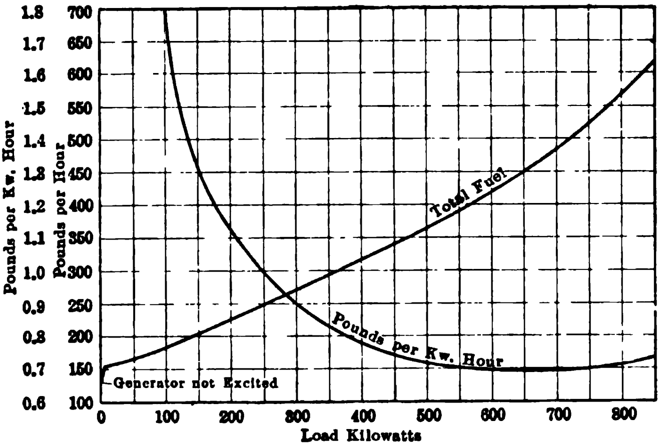

FIG. 4.—AVERAGE FUEL CONSUMPTION TESTS ON DIESEL ENGINE, BURRO MT. COPPER CO.

Diesel Engine: 5 cylinder, 1250 hp. (sea-level rating), 2 cy. ch.
Plant Elevation: 5,950 ft.
Fuel: 16° Bé California crude oil, 18,360 B. t. u. per pound.

The labor cost is high because we have operated from the first with the full labor needed for operation when the power house is delivering the power necessary to run the mines and mill at the rate of 1,500 tons per day, a condition that has not yet been reached.

The repair cost is high, for soon after operations were started the construction account was closed and several items, such as labor for installing recording thermometers, measuring gages, some lubricating-

TABLE 1.—*Burro Mountain Copper Co., Cost of Power*

1915	Labor	Fuel Oil	Lubricating Oil	Miscellaneous Supplies	Repairs	Total	Total, Kw.-hr.	Available Kw.-hr.	Cost per Kw.-hr. Available	Pounds Fuel Oil per Kw.-hr.	Load Factor	Average Load
Jan.	$602.15	$754.79	$98.50	$34.50	$63.93	$1,553.87	101,315	93,322	$0.01665	1.29	0.18	136
Feb.	796.60	742.50	72.60	33.31	198.44	1,843.45	77,400	69,512	0.02683	1.68	0.15	115
Mar.	782.72	859.50	80.06	50.71	179.18	1,952.71	143,820	135,025	0.01446	1.02	0.26	192
Apr.	731.83	919.71	71.45	85.67	291.32	2,099.98	134,473	125,795	0.01669	1.13	0.25	187
May	740.86	959.00	128.86	28.26	456.64	2,313.62	130,690	122,703	0.01880	1.24	0.24	176
June	705.83	882.92	142.96	29.47	393.57	2,154.75	152,646	145,347	0.01482	1.00	0.28	212
July	728.01	1,043.44	111.90	51.56	443.64	2,378.55	159,742	151,089	0.01574	1.16	0.30	228
Aug.	762.68	1,037.96	110.99	34.08	477.01	2,422.72	155,800	149,397	0.01622	1.14	0.28	209
Sept.	673.05	1,046.87	122.77	15.76	507.69	2,366.14	185,753	177,853	0.01330	1.00	0.34	253
Oct.	705.01	1,284.25	122.66	17.97	689.58	2,819.47	275,836	267,174	0.01055	0.81	0.49	370
Nov.	692.50	1,393.87	128.69	40.31	363.06	2,618.43	288,394	280,323	0.00934	0.84	0.53	400
Dec.	702.45	1,492.40	132.50	9.81	316.58	2,653.74	335,702	326,580	0.00813	0.77	0.60	450
.....	8,623.69	12,417.21	1,324.48	431.41	4,380.64	27,177.43	2,141,571	2,044,125	0.01330	1.00	0.33	244
1916												
Jan.	$712.15	$1,527.09	$124.88	$30.60	$285.79	$2,680.51	349,154	339,822	0.0078880	0.764	0.63	470
Feb.	694.43	1,671.94	127.57	21.66	264.84	2,780.44	353,119	344,553	0.0080697	0.750	0.70	520
Mar.	791.45	1,399.03	135.93	20.60	497.61	2,844.62	370,790	362,149	0.0078548	0.750	0.68	505
Apr.	768.65	1,897.18	174.38	21.07	574.02	3,435.30	451,590	441,557	0.0077799	0.780	0.58	433
May	961.97	2,522.00	186.66	29.14	1,024.58	4,724.35	559,550	546,282	0.0086635	0.837	0.57	425
	3,928.65	9,017.24	749.42	123.07	2,646.84	16,465.22	2,084,203	2,034,363	0.0080935	0.780	0.62	464

oil piping, etc., were charged to repairs. During the first year we also took apart the pistons, valves, etc., of the engine to satisfy ourselves that everything was going right and to determine if possible at what intervals this work would have to be done. This is shown by the relation of repair labor to material used in the repairs. The third engine is being furnished with scavenging valves having removable seats, and we decided to alter the valves of the two engines already erected while we needed only one engine in service.

The cost of repairs is shown in Table 2.

The repairs to circulating pumps and water piping are repairs outside of the engine water piping; the repairs on lubricating-oil system are also the repairs outside of the engine. The item of other repairs covers all repairs to building, switchboard, lighting, fuel-oil system and rotary converters. The latter should not be included under the cost of alternating-current power, but to separate them would have involved a great deal of work and the final result would not be appreciably different.

TABLE 2.—*Burro Mountain Copper Co. Power Plant Repairs*

1915	Engines		Circulating Pumps and Water System		Lubricating-Oil System		Other Repairs	
	Labor	Material	Labor	Material	Labor	Material	Labor	Material
Jan.	$46.40	$5.21	$1.50	$2.82	1.10	.....	6.90	
Feb.	142.95	2.61	36.10	2.49	.....	.....	11.70	2.59
Mar.	144.30	7.67	4.60	.....	.....	0.78	18.85	2.98
April	171.05	35.46	2.70	14.08	0.35	0.48	48.25	18.95
May	366.60	6.80	26.70	7.85	5.15	.....	42.30	1.24
June	297.50	3.52	53.80	20.71	2.58	2.58	14.50	0.96
July	234.60	3.51	79.60	58.56	.....	.....	50.10	17.27
Aug.	303.55	70.65	63.80	6.56	12.00	1.10	12.95	6.40
Sept.	223.75	7.67	70.30	40.47	33.95	11.79	95.95	23.81
Oct.	280.80	261.37	61.15	16.43	16.85	6.15	33.30	13.53
Nov.	299.67	5.92	7.95	3.04	2.75	0.15	18.70	24.88
Dec.	229.75	51.85	21.00	0.05	0.95	0.43	8.80	3.75
Jan.	169.00	0.34	19.70	0.10	59.45	9.54	12.20	15.46
Feb.	167.75	3.68	17.35	2.39	14.25	0.44	21.40	37.58
Mar.	284.95	152.50	.....	.....	9.40	2.56	36.05	12.15
Apr.	74.30	3.33	.....	.....	142.00	120.43	122.55	111.41
May	320.95	483.38	8.15	0.47	80.70	29.45	52.80	48.68

The water used in cooling the engines is mine water, which apparently has no bad effect on cast iron, but has a strong action on steel, so that all of our oil filters had to be rebuilt.

The circulating pumps come in for a large repair item due to breakage of check valves above the pumps which fell in and wrecked the pumps.

The two largest items of repairs on the engines were, one main bearing that was drained of its oil by a mistake of the oiler the first night we operated and one high-pressure air-compressor head that was wrecked by starting the engine without draining the air cylinder, after it had been idle some weeks. A small water leak had filled the cylinder with water and the head broke at the first revolution of the engine.

The greatest trouble we have had, not due to carelessness, is with the helical gears driving the camshaft. These gears have a tendency to get loose and one of them split from the keyway. We have not ascertained the exact cause of this trouble, which is now much reduced since iron bands were shrunk on the hub of the gears.

Two important items of running expense were doubtful when the engines were purchased: One was lubricating oil and the other the maintenance cost. The first one is now settled and we hope to reduce the cost from that shown in the table by the use of separate pumps to each point where the oil is delivered in the cylinders, insuring a definite quantity of oil. The present engines have one pump delivering to four points and it happened once that one of the oil pipes became plugged and gave slight trouble on one piston.

The maintenance is still in doubt, and cannot be determined until we reach and maintain regular operations for some time; but the writer thinks it will show a decrease from the figures shown.

The water-cooling connections to the pistons were expected to give some disagreeable troubles, but we have had practically none.

The operation of the engines in parallel is very satisfactory even when the load is light. In actual operation, with one engine in service, the peak load carried has been higher than was expected to be possible. Under test, 900 kw. was the limit with water-rheostat load, but in service, peaks up to 1,000 kw. have been carried for several seconds without seriously slowing down the engine.

On the whole, the operation of the plant has been satisfactory and the costs obtained are considerably better than with a steam plant of the same capacity run under the same conditions, amply justifying the extra investment for this type of engine where the cost of fuel oil is high.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Basic-Lined Converter in the Southwest

BY L. O. HOWARD, GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

WHAT was perhaps the first attempt at basic converting in the Southwest was made by the late Charles F. Shelby at Cananea early in 1907, when he removed the acid lining from one of the 8 by 12-ft. barrel-type converters then in use at the reduction works of the Cananea Consolidated Copper Co. and substituted a lining of one course of magnesite brick. In this shell, he blew 21 taps (about 180 tons) of matte to white metal before the brick gave out along the tuyère line. In every case, the white metal so made was transferred and blown to copper in other converters. The cost of the lining was $700 and Mr. Shelby gave up the experiment as impracticable. No further serious effort to bessemerize copper mattes in a vessel having a more or less permanent lining of basic brick was made by any of the large copper smelters in the Southwest until some months after Messrs. Peirce and Smith had brought the question permanently before the public, and had proved it to be a metallurgical and financial success by their work at Baltimore and Garfield.

Early in 1911, the Cananea Consolidated Copper Co. had in operation at its reduction works at Cananea, Mex., several 8 by 12-ft. barrel-type converters lined with magnesite brick which had previously been operating with acid linings. In the 18 months following, these converters produced 50,000 tons of copper. In June, 1912, this company installed the first of its new equipment of six stands of electrically operated 12-ft. Great Falls type converters, and shortly afterward discontinued the use of the smaller shells.

In September, 1911, the Consolidated Kansas City Smelting & Refining Co. had installed at its El Paso plant two 10 by 26-ft. Peirce-Smith converters. Later, in March, 1913, one standard Great Falls type converter was added.

About the time the Peirce-Smith converters were going in at El Paso, the Calumet & Arizona Mining Co. was experimenting with basic linings in 7-ft. by 10-ft. 6-in. barrel converters at its plant in Douglas, Ariz. These efforts were sufficiently successful to enable the plant to handle the total output of copper in these small shells until the standard Great Falls converters were substituted in June, 1914.

During the year 1911, the Anaconda Copper Mining Co., at its Great Falls plant, proved the worth of the so-called Great Falls converter. It was to this type, now being recognized as standard, that the Copper Queen Consolidated Mining Co. turned when it was decided to equip the reduction works at Douglas with basic-lined converters. In April, 1912, the first of these shells was put in operation. A second was added during the following month, and these two were quickly followed by six others.

In the meantime, the Consolidated Kansas City Smelting & Refining Co. had in operation, at its Hayden, Ariz., plant, two 10 by 26-ft. Peirce-Smith converters and later, in July, 1913, added one standard Great Falls converter.

The Arizona Copper Co. first used the basic-lined converter in October, 1911, at its old plant at Clifton, Ariz. The acid lining in some of the small converters was removed, and one of magnesite brick was substituted. The use of these small converters was continued until October, 1913, when the new plant with its equipment of three stands of electrically operated standard Great Falls converters was put in operation.

By January, 1913, the Old Dominion Copper Mining & Smelting Co. at Globe, Ariz., had installed a 12-ft. Great Falls type converter. It was the intention to handle in this shell the total output of copper, amounting to possibly 36,000,000 lb. yearly, but later, to insure sufficient converting capacity in case of accident to the larger shell, two of the old 7-ft. by 10-ft. 6-in. acid converters were lined with magnesite brick. It was not the intention, nor has it been the practice, to use these small converters unless absolutely necessary, and to date they have together turned out somewhat less than 1,500,000 lb. of copper.

The Detroit Copper Mining Co. at Morenci, Ariz., with a yearly copper output of about 11,000 tons, has never considered it advisable to adopt the large basic-lined converter. Instead, the 7-ft. by 10-ft. 6-in. acid-lined shells have been relined with magnesite brick and with these some remarkable work has been done, despite the serious difficulties that accompany the bessemerizing of copper mattes in such small basic-lined vessels. A record of over 3,100,000 lb. of copper converted with one lining, before any patching of the brick was necessary, has been made.

The Consolidated Arizona Smelting Co., at Humboldt, Ariz., likewise has continued in the very successful use of the small basic-lined converter, but contemplates the purchase of a large unit in the near future.

The International Smelting Co. at Inspiration, and the United Verde Copper Co. at Clarkdale, Ariz., have installed the standard Great Falls electrically operated converters in their new plants, both of which were put in operation during 1915. Unfortunately, owing to the newness of the latter plant and the fact that a proper working basis had not yet been

TABLE 1.—*Operating Data of Basic-Lined Converters in Southwest*

	1	2	3	4	5	6	7	8	9	10
Blast pressure in pounds	13	13.2	14	12	12.4	12–15	10–14	14	11.3	13
Cu. ft. air per minute per converter	5,445	7,691	10,600	4,283	6,054	7,250	......	8,181	4,033	4,117
Cu. ft. air per ton bullion produced	152,449	178,411	141,000	228,116	271,577	137,582	118,328	225,674	245,975	370,566
Cu. ft. air per ton iron and sulphur oxidised	96,145	142,224	156,000	98,847	139,361	112,950	118,328	119,475	240,937	123,583
Oxygen efficiency, taking into account total iron and sulphur oxidized	84.0	54.1	57.2	76.8	56.3	74.0	81.0	50.6	69.8	59.0
Total tons iron and sulphur oxidized per stand per month	2,075	1,610	......	1,647	1,903	2,868	1,991	2,576	768	1,440
Punchers used per shell per shift	2	2	2	2	2	2	2	1	1.5	2
Tons bullion per puncher per month	221	204	522	119	478	392	255	137	129	240
Tons iron slagged per puncher per month	192	153	....	169	533	251	171	156	147	
Average time of blow, hours and minutes	4–00	4-11	3–09	6–44	9–20	8–00	8–30	7–12	4–51	10–00
Average time to blow one ton bullion, minutes	28	23	14	54	45	19	28	28	51	90
Average tons bullion per blow	8.5	10.8	13.3	8.22	12.7	25.3	18.0	15.6	5.8	6.7
Average weight of matte per charge, tons	21.9	26.0	25.9	33.9	40.5	60.0	50.0	51.2	19.7	26.6
Tons ore charged per ton of matte	0.23	0.20	0.155	0.201	0.17	0.25	0.30	0.21	0.286	0.283
Tons iron slagged per ton of available silica	......	2.44	......	2.45	3.85	1.75	1.80	2.36	2.03	2.60
Size of converters	12 ft.	12 ft.	12 ft.	12 ft.	12 ft.	10 by 26 ft.	12 ft. Gt. Falls 10 by 26 ft. P.-S.	12 ft.	7 ft. by 10 ft. 6 in.	9 ft. by 7 ft. 6 in.
Number of tuyères	28	24	24	24	24	35	24 Gt. Falls. 35 P.-S.	22	12	12
Size of tuyères, inches	1¼	1¼	1½	1½	1¼	1¼	1¼ Gt. Falls. 1½ P.-S.	1½	1½	1¼
Thickness of brick used on tuyère line, inches	24	30	30	14	30	18	18	24	18	15
Average slag analysis:										
SiO_2	19.6	21.7	17.2	17.4	15.6	23.6	25.7	29.2*	22.2	23.8
FeO	68.2	67.9	69.1	70.7	67.3	66.2	60.0	62.1	63.4	61.5
CaO	0.8	1.1	0.6	0.8	0.4	......	......	1.6	0.9	
Al_2O_3	3.4	3.0	3.7	3.0	2.9	......	......	....	2.8	
Average matte analysis:										
Cu	37.7	43.37	51.2	28.22	34.7	43.7	42.8	31.9	35.21	25.0
Fe	32.5	29.2	21.6	30.9	34.9	28.0	27.4	28.8	36.1	46.0
S	27.4	23.5	24.5	25.3	24.9	25.1	25.6	18.7?	25.0	25.0

1. Arizona Copper Co., Clifton, Ariz.
2. Old Dominion Copper Mining & Smelting Co., Globe, Ariz.
3. International Smelting Co., Inspiration, Ariz.
4. Calumet & Arizona Mining Co., Douglas, Ariz.
5. Copper Queen Mining Co., Douglas, Ariz.
6. Consolidated Kansas City Smelting & Refining. Co., Hayden Plant, Ariz.
7. Consolidated Kansas City Smelting & Refining Co., El Paso Plant, Texas.
8. Cananea Consolidated Copper Co., Cananea, Sonora, Mex.
9. Detroit Copper Mining Co., Morenci, Ariz.
10. Consolidated Arizona Smelting Co., Humboldt, Ariz.

* Insoluble.

arrived at, it was impossible to obtain any figures covering the converting operations.

There is no question about the superiority of basic over acid linings or, where the output of copper is sufficiently large to insure efficient operating, of the standard type over the smaller basic-lined converter. The inability properly to control the temperature of the charge in the small vessel, and the resulting destruction of the brick lining, are the most serious difficulties with which the operator using a small basic-lined converter has to contend. Throughout the Southwest, the preference has been for the standard upright rather than for the horizontal converter. The greater flexibility of the former, and the greater ease with which repairs may be made, strongly recommend it.

The operating data in Table 1 will give an idea of the working conditions and the results obtained at some of the large smelting plants in the Southwest. It is to be regretted that replies to questions pertaining to power consumption were at such variance that it was thought best to omit them entirely, as no reconciliation seemed possible.

TABLE 2.—*Old Dominion Copper Mining & Smelting Co.*

Campaign of basic-lined converter No. 2.	
Put in operation	June 27, 1913
Removed for initial patching	Dec. 7, 1915
Total hours blowing	13,734
Total number blows made	3,288
Average time of blow	4 hr. 11 min.
Total number taps matte	9,316
Total tons matte charged	85,578
Average weight of matte per charge	26 tons
Average copper content of matte	43.37 per cent.
Total tons bullion produced	35,431
Average time to blow one ton copper	23.2 min.
Average tons copper per blow	10.80
Average blast pressure	13.2 lb.
Average air used to blow one ton copper	178,411 cu. ft.
Average air used per minute	7,691 cu. ft.
Average air used per ton iron slagged	242,502 cu. ft.
Total ore fed	17,097 tons
Tons ore fed per ton matte blown	0.200
Magnesite brick used for repairs	None

Average Slag Analysis	Per Cent.	Average Matte Analysis	Per Cent.
Cu	2.1	Cu	43.37
SiO_2	21.7	Fe	29.2
Fe	52.2	S	23.5
CaO	1.1		
Al_2O_3	3.0		
S	1.8		

Separate operating data for the Peirce-Smith and Great Falls converters at El Paso could not be obtained, so the figures in column 7 are a combination of results obtained in these different types. The figures in columns 9 and 10 are for former acid-lined shells now operating with a lining of magnesite brick. All figures are for periods of 1 month or more. Those in column 5 are for the year 1915. The data in column 2 cover a period of 29 months' operation—the life of one original lining to the time of first patching. Although some of the figures in Table 2 have already been given, this more complete summary of the campaign may prove of interest.

I am greatly indebted to the managers and superintendents of the different properties who so graciously and promptly furnished me with the necessary information for this article, and take this opportunity to thank them.

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Mine-Fire Methods Employed by the United Verde Copper Co.

BY ROBERT E. TALLY,* B. S., JEROME, ARIZ.

(Arizona Meeting, September, 1916)

UNDERGROUND fires have been common in the mines of the United Verde Copper Co. for the past 22 years. The first fire started in the 300 Hampton stope in the fall of 1894, following a cave in that orebody. The soft and highly pyritic nature of the ores is responsible for most of these fires. Since ore characteristics, methods of mining and ventilation, have an important influence on mine fires, they will be described in this paper.

Orebodies

The ores are mostly chalcopyrite except on the upper levels where considerable chalcocite, black oxides and other secondary minerals occur. The orebodies are in the form of lenses and vary in size. Some are small, while others are several hundred feet in length by a hundred or more feet in width. These orebodies extend, in some cases, from the surface to the lowest levels of the mine and usually at a steep angle of inclination.

There are three different classes of ores, classified in accordance with the gangue rock. The copper in all cases, except as above specified, is in the form of chalcopyrite. The largest and most important of these reserves are in a pyrite gangue. Next in order of importance are the schist ores and lastly the quartz porphyries. The pyrite, locally termed iron ores, average from 30 to 42 per cent. sulphur. The schist and quartz porphyries vary from 3 to 25 per cent. sulphur.

Mining Methods

All the orebodies were worked by the overhand square-set method of mining prior to 1908; since then the cut-and-fill, shrinkage and mill-hole systems have been largely employed.

But 15 per cent. of the total tonnage now extracted is from timbered stopes. Most of this is from the upper levels where the ore is soft, heavy and more or less broken to the surface. Large reserves of these ores are being conserved for more efficient methods of mining.

---

* Superintendent of Mines, United Verde Copper Co.

There was no choice in the past with reference to the methods of working the upper levels because the smelting plant and other surface buildings were directly over the orebodies. Only such places were worked as were considered necessary and by methods which caused as little settling as possible. In most of these places it was necessary to use spiling and to fill to the roof. Bulkheads were used on the levels to retain the gangways and were carried up through the stopes for ore chutes.

Top slicing could not be used on account of surface settling and because this method is unfavorable, in heavy sulphide orebodies, for the control of mine fires.

The smelting plant and other surface interferences have now been removed and it is planned to start steam-shovel work in the near future.

History of United Verde Fires

The first fire of 1894 was caused by spontaneous combustion, following a cave. It was not extinguished and the district was bulkheaded from the remainder of the mine. The soft and broken nature of the ground in this district was such that the smoke and gas worked its way over the orebody on the several different levels and cut off a large productive area of high-grade ore.

Later, another serious fire occurred in the Chrome or quartz-porphyry orebodies, which extended over a large area and cut off production from this district.

A third fire followed, cutting off the middle district which lies between the Chromes and Hamptons. Many smaller fires occurred in the meantime, which were extinguished. There has never been a serious fire in the schist orebodies which join the Hamptons on the south, yet there is always danger in that or any other heavily timbered district, where the ore carries any appreciable amount of sulphur.

Fire Stopes

Many attempts were made to extinguish these fires and recover the large reserves of high-grade ore. The districts were first flooded with water but, owing to the large area involved and the soft, broken-up condition of the ground, this method was a failure. Sufficient air always found its way through fractures to feed the fire.

The next attempt to extinguish the fire was by CO_2 gas. A gas-generating plant was installed, pipe lines were laid and large volumes of gas were forced into the district, but without effect. Steam was then tried and was likewise a failure. Water, steam or CO_2 gas will readily extinguish a fire if the district in which the fire occurs can be tightly sealed up and the extinguishing agent brought into contact with the fire in sufficient quantities. Fractures to the surface and underground workings outside the fire district made an impossible condition to seal up.

In 1905, the Plenum system was tried on this district with success. Actual combustion was extinguished and the latent heat reduced to a temperature of about 120°. This work was planned and directed by

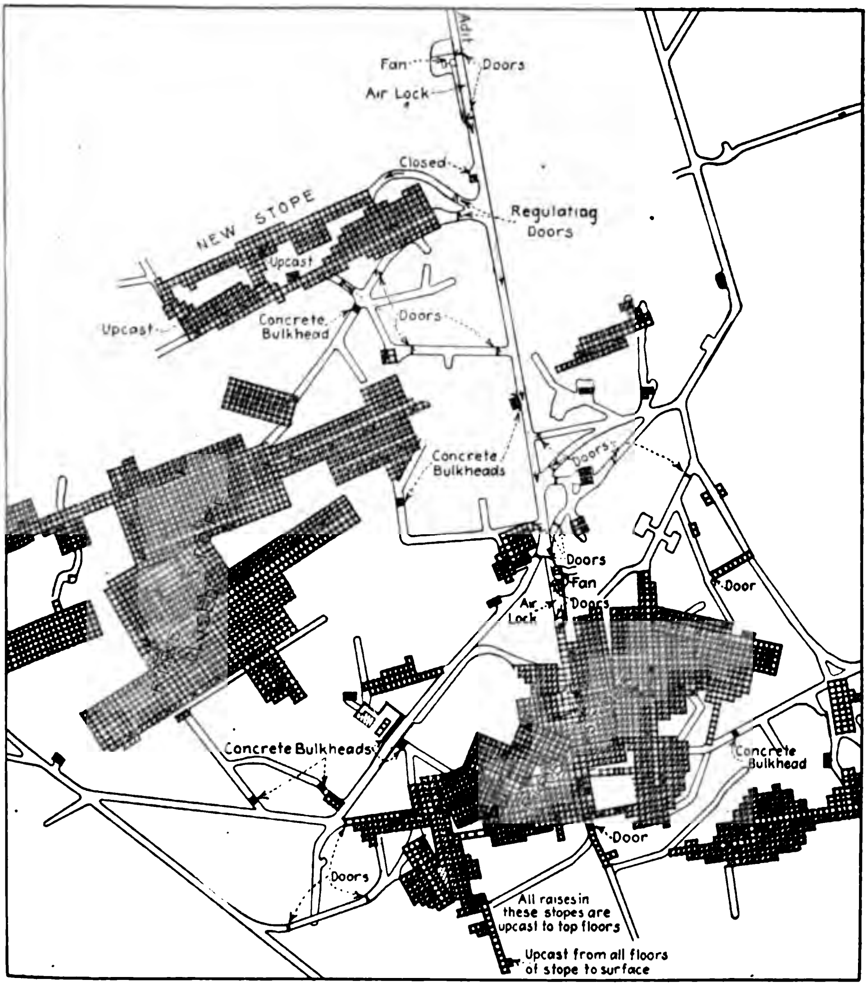

FIG. 1.—PLAN OF FIRE STOPES ON 300 LEVEL SHOWING METHOD OF VENTILATION.

Joseph J. Shaw, a mining engineer from the Mountain Copper Co. of Keswick, Cal., where this system was successfully used. It has been used here continually since its inception and consists in forcing the air under a pressure, varying from 2 to 5 lb., into the fire district. The air pressure varies in accordance with the gas pressure and must be sufficient to keep back the gas and to cool the ground, so that work can be accomplished.

When a bulkhead, in a connection leading to a fire stope, is opened there is invariably an outward pressure and rush of gas or smoke, which must be overcome. The Plenum system consists in forcing the air against this gas at a pressure slightly greater than that of the gas.

Fig. 1 shows a large fire stope on the 300 level, with doors for controlling the pressure, and the method of ventilation. The air is furnished from the adit and forced under pressure into the fire district. All chutes and manways are upcast, and the spent air and gases outlet to the air raise through connections on the different floors.

The installation used with this system consists of a No. 8 double-inlet Sirocco fan, direct-connected to a 50-hp. variable-speed General Electric motor; also the necessary airways, doors, etc. The air from the fan must be under perfect control in order to regulate the gas. This is accomplished by doors, as shown in Fig. 2.

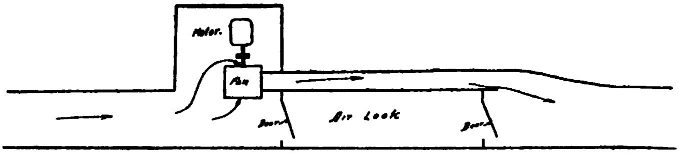

FIG. 2.—SKETCH OF FAN STATION.

An airway about 3 ft. square and from 50 to 200 ft. long is connected to the outlet of the fan. Doors are placed in the tunnel at both ends of this airway. These doors prevent the fan from drawing its own air, for when one door is open the other must be closed. All workings between the fan and the fire district must be closed off by doors; otherwise the air pressure will be reduced and trouble result from gas. Two doors are always used so that one will remain closed when the other is open. For electric haulage these doors should be from 100 to 200 ft. apart. The doors are made of 2-in. grooved lumber, and concrete frames. The airways are made of lumber or concrete.

With a variable-speed motor and the proper system of doors, it is an easy matter to increase or diminish the air pressure in order to meet any new conditions within the fire district.

When the bulkheads to this district were first opened, the ground was red hot. The roof and walls, where opened, were aglow. All timber was burned and the soft sulphides, where exposed, were burning. The continual blowing of air on this hot mass gradually cooled it, so that within a few weeks the actual extraction of ore was under way.

The timber often took fire before it was blocked, making it necessary to keep it well sprinkled with water. The air reduced the temperature from 1,200° F. to 120° F. in about 6 weeks. The conditions gradually improved until the average temperature in the fire stopes was 100° F. During recent years a system of ventilation has been introduced whereby the temperature has been reduced to about 75°, except in places that are above the line of ventilation.

Formerly the air was forced into these stopes and found an outlet through fractures into the other workings and to the surface, with the result that none of the upper levels were free from gas.

A system of air raises was driven in the foot wall to the surface and connections from these raises to the stopes were made at different elevations. Doors were placed in these connections and the air outlet thereby controlled. If an outlet is too free, it will not only release the pressure and cause gas, but, by suction, will also cause fire. These raises were driven in ground that did not require timbering. Cast-iron sets were used where necessary in the connections from the raises to the stopes. Timber should be avoided if possible in airways, since there is a constant danger of fire in these places.

These stopes have now been opened for 11 years and fire occurs on an average of about once a month in a rather extensive area of old timbered workings. It is therefore important to have an efficient fire-fighting organization and to have good control of the ventilation. This is an important factor in connection with mine fires. In the first place, every preventive measure must be used, and, in the second, the ventilation must be under absolute control in order to extinguish the fire.

An excess of air increases the flames in the same manner that an open draft increases the fire in a stove or furnace. Air must be furnished only in sufficient quantities to carry away the smoke so that the firemen can get at the fire. Where the smoke is thick it is impossible, even with helmets, to handle the fire satisfactorily. Many unsatisfactory results in connection with mine fires are due to these causes.

Causes of Mine Fires

Probably 90 per cent. of all underground fires in pyrite or heavy sulphide ores are caused by spontaneous combustion. In the fire district of this mine, many fires are due to red-hot dust dropping on the timbers, from fractures in ground above. Some fires are from friction due to caving, some from defective electric wires or cables; others from burning powder in contact with timber, and a certain number from incendiarism and carelessness with candles, lamps, etc.

Fires from spontaneous combustion usually occur in the interior of a gob or filled stope and are due to the oxidization of fine sulphides in contact with timber. Very small amounts of air are necessary for spontaneous combustion and this air works its way through the filling along the line of bulkheads, timber or walls, where it comes into contact with sulphide dust. In the process of oxidization this produces sufficient heat to start a fire. This theory has been proven here by experiments. Heavy sulphides should not be used for filling in timbered stopes. Considerable fine sulphide ore is lost in the fire stopes or in any other heavy ground where the square-set method of mining is used and where the floors are kept filled to the roof.

Prevention of Mine Fires

Careful consideration should be given toward making heavy sulphide mines as fireproof as economy will permit.

Main hoisting shafts in such properties should be constructed of concrete or other fireproof material. Where timber is used, a sprinkling system should be installed.

Mine methods should be developed in which timber will be eliminated as far as possible.

Heavy sulphide material should not be used for filling in the timbered stopes. In these places fine waste should be used and sprinkled so that it will pack and make conditions unfavorable for fire.

In fire stopes or stopes adjacent to the fire district, the timber next to the walls or ends should be removed.

Careful attention should be directed toward ventilation so that the temperature will be unfavorable for fires to start.

In hot places the timber should be kept damp by sprinkling with water. Water should never be put on red-hot or burning ground, as this will invariable cause an explosion, one of the great dangers in handling mine fires.

Water and air lines, hose and hose connections and all tools used in connection with mine fires should be kept convenient and in good working condition where fire is liable to occur.

Helmets, pulmotors or lungmotors, electric lamps and supplies should be kept in stock in sufficient quantities and convenient for immediate use.

Carefully selected men should be trained in the use of the helmets and resuscitating machines, and in the methods of fighting fires. The average man, unless specially trained, is useless in fighting fires. The foremen, shift bosses, electricians, pumpmen, watchmen and selected miners working in different places which are subject to fires should familiarize themselves with all things appertaining to the ventilation, and should know just what to do if fire occurs in their respective places. Rules to this effect should be printed, distributed, and understood by all concerned.

Watchmen should inspect the air outlets often and regularly in order to detect smoke as soon as possible.

Iron fire doors should be erected in connections near all timbered stopes, so that in case a fire gets beyond control the doors can be closed and the district quickly sealed up, in order to protect the remainder of the mine. Similar doors should also be erected near timbered shafts, airways, etc., for protection against fires.

A fire signal should be in use at all mines and understood by all employees. A practice signal should be given occasionally. The following fire signal is in use here:

> "In case of fire, ring nine bells on the electric cage call signal. The engineer on receiving these bells will flash all electric light throughout the mine, nine times. This flash will be repeated three times and followed by flashing the station signal on which level the fire is, three different times. Carmen and all others working where there are electric lights will notify those employed in stopes and other places where there are no lights. The trained firemen on the various levels will then take charge of the situation. Their first consideration will be for the safety of the men and then for the extinguishing of the fire and protection of property."

When a fire occurs in the fire district no signal is given, for the system there is such that the fire can be easily confined without danger from smoke or gas to the other parts of the mine.

When fire occurs in any other part of the mine, the signal is given and all men working where there is danger from suffocation are instructed to go to the surface or other places of safety. The next step is to get water on or over the fire as quickly as possible. When the fire is in the interior of the gob, it is sometimes necessary to drift from a chute or manway through the filling in order to get at it. Often, however, the fire can be extinguished by water from above. This is usually accomplished by driving holes in the filling with a pinch bar and turning water into these holes. For this work, hose nozzles made of 1-in. pipe with about a 3/8-in. hole are used. By this method, water can be scattered all over the stope. It will keep the fire from spreading and usually extinguish it.

In case a fire gets beyond control, the fire doors should be immediately closed, after which concrete bulkheads should be erected. It is sometimes possible to extinguish a fire in a sealed-up stope by water, through a diamond or churn drill hole. In sulphide stopes, however, it is usually necessary to burn the timber before the stope can be recovered. A fire, when apparently extinguished, may start again when the bulkheads are opened, unless the timber is consumed.

At the present time one large fire district here is being experimented with along these lines and with apparent success. Air in small amounts is gradually forced into the stopes to burn the timber. This causes the filling to settle, and, to avoid caving, more waste is put in from above.

Ventilation

Ventilation has received considerable attention here during the past 10 years, not only in the fire district but throughout the entire mine. Sulphide mines are naturally warm and disagreeable unless ventilated. Natural ventilation is used where possible; otherwise, fans are used to blow the air where it is required. The ventilation of the fire stopes interferes with the remainder of the mine, since the air is taken from its natural source and forced into the fire district. Artificial ventilation is used for the middle and lower levels.

Fig. 3 is a plan of one level showing the location of the fan and the method of ventilation. All levels are connected to the air raise.

Fig. 4 is an ideal vertical section of the orebodies, showing the direction of the air currents, and the methods of ventilation.

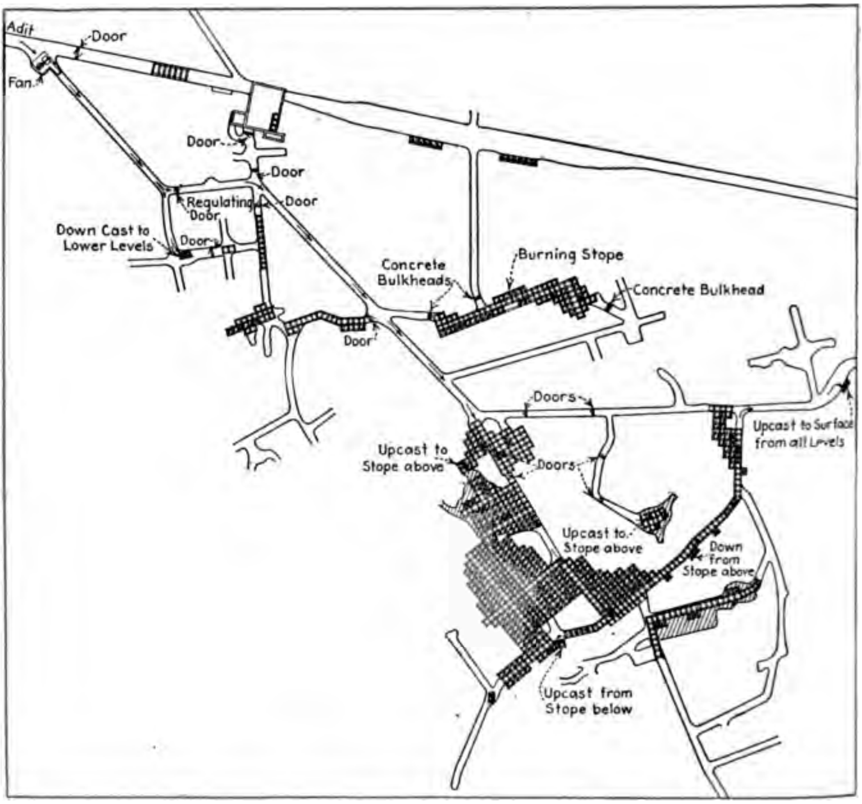

FIG. 3.—PLAN SHOWING LOCATION OF FAN AND THE METHOD OF VENTILATION.

Fans are installed in suitable places where they can draw air from the outside and force it into the different workings, thence to the air raises to the surface. The same airways and doors are used as for the fire district, except that no particular pressure is maintained, the air having as free an outlet as possible to the surface. The ventilation for any particular place or section is regulated and controlled by doors. For the proper ventilation of stopes, the air should have an inlet at one end and an outlet at the other. Most air is forced into the main working places and only a sufficient amount is sent to the old workings to preserve the timber and make conditions unfavorable for fire.

The maximum temperature in the middle and lower levels is 80°F. It varies from 60 to 80° F. and averages about 72°, with small amounts of humidity.

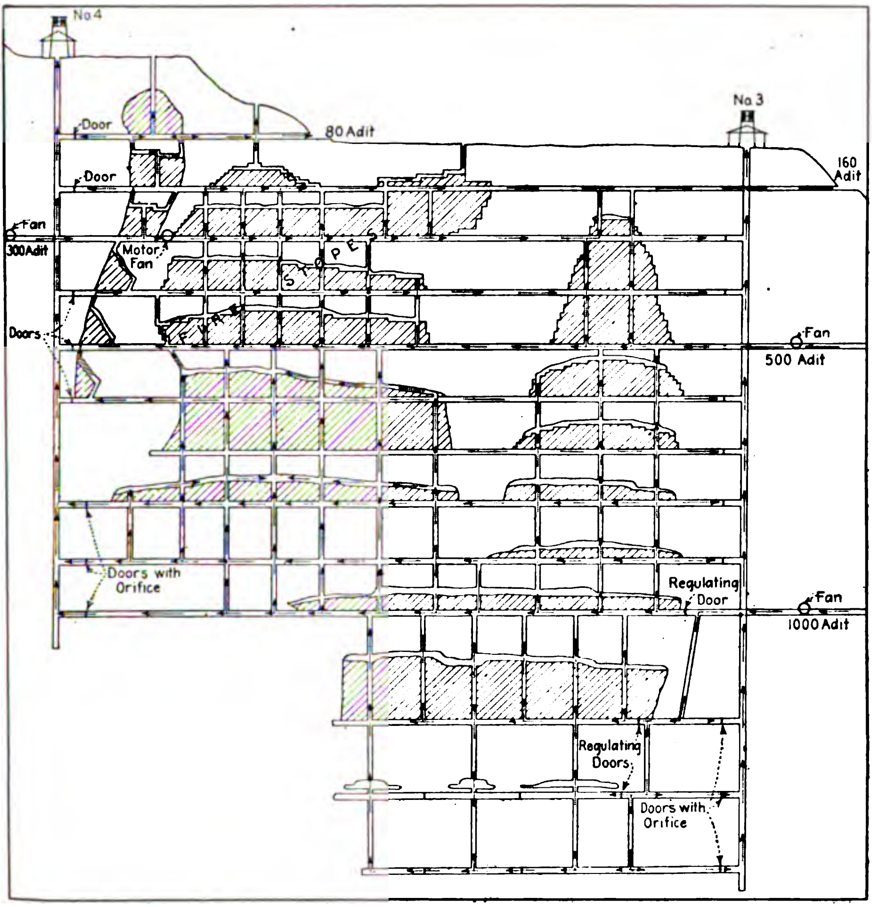

Fig. 4.—Vertical Section Showing Ventilating System of United Verde Mine, Jerome, Ariz.

Conclusions

For handling mine fires satisfactorily, the Plenum system as employed here has been a marked success, but the mining methods in connection therewith are necessarily expensive and inefficient.

Mine fires are the most disagreeable and dangerous conditions that are encountered in metal mining and every possible precaution should be exercised to prevent them. In addition to preventive measures, there should be carefully trained firemen who thoroughly understand how to control and extinguish fires.

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Cyaniding Clayey Ore at the Buckhorn Gold Mine

BY PAUL R. COOK,* B. S., GUAYAQUIL, ECUADOR

(Arizona Meeting, September, 1916)

THE ore deposit of the Buckhorn Mines Co., Buckhorn, Nev., is peculiar in being a shallow kaolinized mass of material with basalt walls, and having apparently no direct connection with any of the usual gold-bearing rocks. The average ore contains 16 per cent. water of hydration, and the cyaniding of this hydrous clayey material offered unusual difficulties as compared with the typical gold quartz ores of Nevada.

The orebody was thoroughly developed; then the mill was built according to the latest cyanide practice, with such changes as were thought to be demanded by the peculiar nature of this ore.

Upon starting the mill, the ore proved more difficult to handle than had been anticipated. It is hoped that an account of how these difficulties were met may prove of interest to anyone having a clayey ore to handle and to the profession in general.

GEOLOGY

The Buckhorn orebody lies along a north and south fault plane of perhaps 1,000 ft. dislocation, that can be traced for miles; but the only other known mineralization consists of similar ore in the Murphy mine, a mile farther north.

The east or hanging wall is hard and smooth, being a typical fault plane. The best ore is along this wall, gradually grading down toward the west until at 30 to 60 ft. it is too low-grade to mine. The country rock on the west consists of alternating layers of hard and soft basalt and basalt scoria, pitching toward the mine.

One of these basalt layers on the hillside a little above the mine is marked, for 3 or 4 miles in length, by a line of springs which seep out along it. Perhaps the meeting of the surface drainage, passing down these basalt layers, with the fault-plane solutions explains the formation of the Buckhorn orebody.

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* With Buckhorn mines during the first year of mill operation (December, 1913, to December, 1914).

Beneath the oxide ore a smaller body of almost pure marcasite occurs with about the same assay value ($8 per ton). Beneath the 250-ft. level there appears to be no further mineralization.

MINING

The first difficulty was to get the ore out of the mine. Since the orebody extended from the grass roots to a depth of 175 ft., with a width of 50 to 80 ft. and a length of 1,400 ft., the "glory-hole" system of mining was adopted. The ore was broken through 8-in. grizzlies into 18 chutes extending from the surface to an intermediate level, on which it was trammed by hand to three chutes leading to the 200-ft. level, and hauled by electric motors through a 1,000-ft. tunnel to the mill.

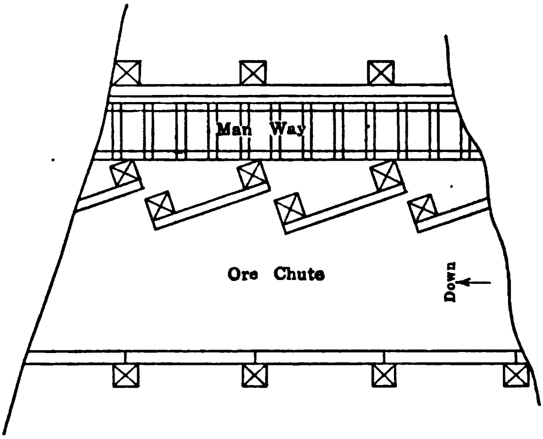


FIG. 1.—SECTION THROUGH ORE CHUTE.

In the summer little trouble was experienced; but the ordinary rain and snowfall of a Nevada winter made it almost impossible to keep the chutes open, as first built. After reconstructing, so as to provide an opening for barring at each set of timbers (see Fig. 1), it became possible to keep the mill supplied with ore during all kinds of weather. During wet weather all the cars had to be scraped with a shovel after dumping, and unusual care was required to keep a sticky load from carrying car and all into ore bin or over the dump; 30 per cent. of the tonnage mined went to the waste dump.

MILLING

Ore Bin and Crusher

The next problem was to get the ore out of the mill bin and crushed. The bin was an ordinary circular steel bin, with natural earth bottom and side gate. This ore absolutely refused to run from the bin. The mill was built to treat 300 tons a day, but even with one man in the ore bin, and two at the crusher, it was impossible to get over 150 tons through in 24 hr. The large kaolin lumps gave the most trouble in crushing. They had to be practically chiseled to pieces and poked through the jaw crusher by hand.

The replacement of the jaw crusher with a high-speed toothed roll (see Fig. 2) gave the desired crushing capacity. This machine was developed at one of the Bingham Canyon (Utah) properties, and is manufactured by a Salt Lake firm. It is well adapted for sticky ores. The Buckhorn ore contains an occasional "nigger head" of very hard "malapi" or basalt. The mill crew was afraid one of these would break off the teeth of the crusher shell, and they were very carefully picked out at first, but after a few had gone through by accident, and a few pieces of drill steel, etc., had been chopped up, no further concern was felt. All the boulders that had been sorted out were later put through as it was found that the easiest way to keep the teeth free of adhering clay, was occasionally to throw in a boulder of hard rock.

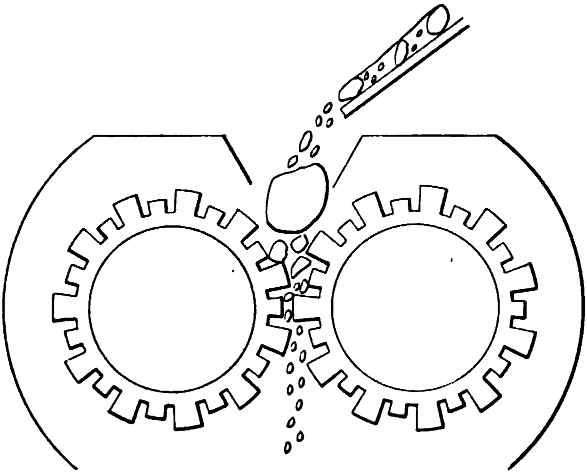


FIG. 2.—SECTION THROUGH WALL CRUSHER SHELLS.

To do away with the necessity of a man on each shift to shovel the ore out of the bin, a 36-in. conveyor belt was installed to feed the crusher automatically. The opening in the bottom of the bin, over the belt, was about 2 ft. wide, and extended clear across the bin. It was closed by means of short pieces of mine rail that could be removed a few at a time, permitting the ore to be drawn from any point desired. The belt was driven from the crusher line shaft by means of ratchet and dogs. It could be started or stopped, from either floor of the crusher building, by means of a rope connected with the dogs. With these improvements, crushing required only a part of one man's time. Two men each shift attended to crusher, rolls, two ball mills, two classifiers and two tube mills.

Rolls

The 45 by 15-in. Anaconda-type rolls with smooth shells would clear themselves fairly well, if one of the shells had a channel about 1 in. wide by ½ in. deep machined in it, but it was troublesome to keep a groove in the shells as they wore down. A corrugated or toothed shell would have been better for this ore.

Ball Mills

One 6-ft. Hardinge ball mill was intended to handle the whole tonnage. After plastering the balls to the side of the mill with clay, a few times, the shiftmen learned to tell by the sound of the mill, when it was beginning to coat up. By shutting off the feed at this time, it took only a few minutes for the coating to be ground out.

The installation of a duplicate ball mill made it possible to keep the rest of the plant going while grinding out the ball mills, one at a time, and allowed the rolls to be set coarser on troublesome ore. With a good run of ore, 300 tons per day was sometimes put through one mill.

Classification and Tube Milling

This ball mill discharge was classified in two 36-in. Akins classifiers, the sand from which was fed to two 5 by 18-in. tube mills with Komata liners. The tube-mill discharge was classified in a home-made drag classifier. The small percentage of material requiring regrinding consisted largely of fragments of the basalt "nigger heads." This material was almost as hard as the pebbles themselves and of low assay value. Occasionally it accumulated in the circuit enough to be troublesome and was thrown away. A small amount of it was thought to help the grinding. About 80 per cent. of the product delivered to the treatment plant would pass a 150-mesh screen.

Agitation

About 80 per cent. of the mill-head value was dissolved in the crusher plant. Only a trifling additional extraction could be obtained either in the mill or experimentally. The real trouble was to remove the dissolved value from the clayey pulp. Accordingly the three 32 by 14-ft. Dorr agitators were changed to thickeners.

THICKENING

These three "converted" agitators settled 300 tons per day of 1 to 10 pulp, as delivered from crusher plant, to a specific gravity of 1.15. The 8 sq. ft. of settling area provided per ton of this ore settled in 24 hr. would be sufficient to settle an average Nevada quartz ore to a specific gravity of 1.33. The overflow was precipitated, and the underflow mixed with the barren solution and fed to six 36 by 12-ft. Dorr thickeners, delivering a 1.23 specific gravity underflow to the filters. The 20 sq. ft. of settling area per ton settled in 24 hr. is three times the area required to settle an average Nevada quartz ore to a specific gravity of 1.33. Primary thickeners were held with 2 ft. of clear solution; the secondary thickeners with 6 in. It was impossible to settle the raw Buckhorn ore beyond a specific gravity of 1.26, either in the mill or experimentally.

Filtering

The maximum capacity of each of the four 14-ft. diameter by 12-ft. face Oliver filters was 50 tons per day, about one-half their capacity on a Nevada quartz ore. An additional filter, 14-ft. diameter by 24-ft. face, had to be installed to handle 300 tons per day.

Dehydration

A sample of Buckhorn ore carefully dried at a temperature below 110° C. had a specific gravity of 1.9. A higher temperature gave an additional loss of 16 per cent. in weight, and entirely changed the physical properties of the ore. The dehydrated sample had a specific gravity of 2.4, and settled and filtered almost as well as a quartz ore. Dehydrating also removed the sticky milling qualities. Both samples, however, gave the same extraction with cyanide. The temperature of a laboratory electric hot plate was sufficient to dehydrate a sample nicely. As CO_2, etc., would not be driven off at this temperature, this loss in weight must be due to water of hydration.

With a cheap fuel supply, dehydration before milling would be the best treatment for this class of material. The ore would mill and classify more easily; the thickeners and filters would have normal capacity; and dissolved values would be more completely removed. The temperature of a commercial drier would dehydrate the ore with about the same fuel consumption (100 lb. of coal per ton of ore) as in removing the 18 per cent. of H_2O if it existed in the form of moisture.

The high price of fuel delivered at Buckhorn prevented the adoption of dehydration at this mill. The ore was milled raw at the cost of $1.59 per ton (see Table 1). Power cost $8 per horsepower per month.

The careful drying at a temperature below 110° C. of a large number of samples, with the equipment of an ordinary cyanide plant assay office, would be a rather tedious operation; so the regular moisture and assay samples at Buckhorn were dehydrated. All assay, moisture, tonnage, etc., figures are on this basis.

To compare with other ores the figures obtained by drying below 110° C. should be used. Both sets of figures are given in Table 2. Table 3 shows the mill flow sheet.

TABLE 1.—*Buckhorn Mining and Milling Costs*

Ore Milled, 10,000 Wet Tons, 8,100 Dry Tons, H_2O 19 Per Cent.

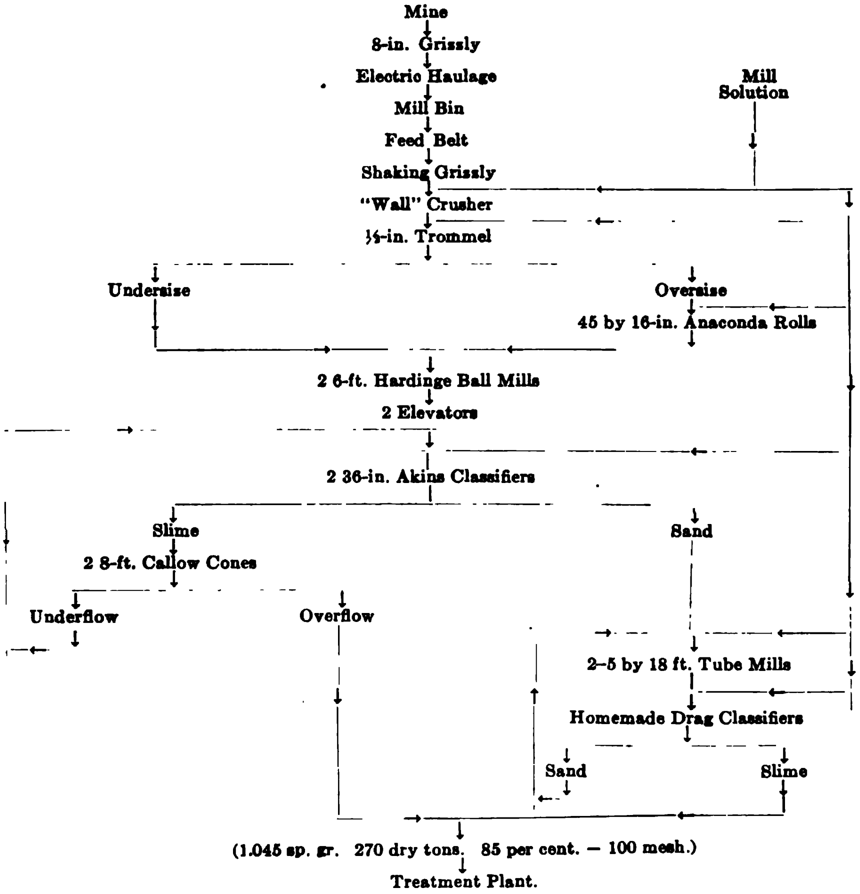
MINING		
Ore Breaking:	Per Ton	
Labor	$0.259	
Supplies	0.098	$0.357
Tramming 100-*ft. Level:*		
Labor	0.044	
Supplies	0.002	0.046
Timbering	....	0.013
Electric Haulage:		
Labor	0.084	
Supplies	0.006	
Power	0.022	0.112
General Expense:		
Surface drainage	0.004	
Haulage tunnel repairs	0.039	
Assaying and sampling	0.027	
Surveyor	0.011	
Superintendence	0.067	
Incidentals	0.024	
Development	0.083	
Overburden and waste	0.174	0.429
Grand Total	....	$0.957
MILLING		
Crusher and Rolls (Wall-toothed Roll and 45-in. Roll):	Per Ton	
Labor	$0.057	
Supplies	0.003	
Power	0.034	$0.094
Hardinge Ball Mills (Two 6-ft. Mills):		
Labor	0.050	
Supplies	0.026	
Power	0.076	0.152

Elevating and Separating (Two 36-in. Akins Classifiers):			
Labor	0.006		
Supplies	0.001		
Power	0.002	0.009	
Tube Milling (Two 5-ft. by 18-ft.):			
Labor	0.011		
Supplies	0.049		
Power	0.092	0.152	
Agitation (Three 32-ft. by 4-ft. Dorrs):			
Labor	0.028		
Supplies	0.006		
Power	0.027	0.061	
Chemicals:			
Cyanide	0.218		
Lime	0.293		
Lead acetate	0.024	0.535	
Continuous Decantation (Six 35-ft. by 12-ft. Dorrs):			
Labor	0.004		
Power	0.017	0.021	
Filtering and Discharging (Six 14-ft. by 12-ft. Olivers):			
Labor	0.062		
Supplies	0.045		
Power	0.088	0.195	
Precipitation:			
Labor	0.025		
Supplies	0.067		
Power	0.019	0.111	
Return Pumping:			
Labor	0.009		
Power	0.012	0.021	
Refining:			
Labor	0.023		
Supplies	0.021		
Power	0.001	0.045	
Assaying and Sampling:			
Labor	0.018		
Supplies	0.008	0.026	
Superintendent and Foremen		0.090	
Experiments		0.010	
General Expense		0.043	
Water Supply		0.021	
Grand Total Milling			$1.586
Grand Total Mining			0.957
Grand Total Mining and Milling			$2.55

TABLE 2.—*Milling Data*

	Figures Obtained by Dehydrating Samples		Figures Obtained by Drying Below 110° C.		Comparative Figure for Quartz, 2.7 Specific Gravity
	H_2O, Per Cent.	2.4 Specific Gravity, Dry Tons	Moisture, Per Cent.	1.9 Specific Gravity, Dry Tons	
Ore milled per month, 10,000 gross tons	19	8,100	4	9,600	2 per cent. H_2O
Ore milled per day, 333 gross tons	19	270	4	320	2 per cent. H_2O
Ball mill discharge, specific gravity, 1.439	48	270	36	320	
Tube mill discharge, specific gravity, 1.394	52	234	40	279	Specific gravity, 1.64; moisture, 38 per cent.
Akins classifier, sand product	38	175	25	208	Moisture, 24 per cent.
Slime to treatment plant, specific gravity, 1.045	1 to 11.5	270	1 to 10	320	1 to 7, specific gravity, 1.08
Primary thickeners underflow, specific gravity, 1.15; settling area, 8 sq. ft. per ton	1 to 3.5	270	1 to 2.57	320	1 to 2, specific gravity, 1.26 or better.
Secondary thickener underflow, specific gravity, 1.23; settling area, 20 sq. ft. per ton	1 to 2.1	270	1 to 1.5	320	1 to 1, specific gravity, 1.46 or better.
Filter cake	43	270	30	320	Moisture, 29 per cent.

TABLE 3.—*Flow Sheet.*



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Possibilities in the Wet Treatment of Copper Concentrates

BY LAWRENCE ADDICKS,* B. S., NEW YORK, N. Y.

(Arizona Meeting, September, 1916)

AT the San Francisco meeting of the Institute last year, I presented, through the courtesy of Dr. James Douglas, some results of experiments on the roasting and leaching of concentrator tailings. After it became apparent that flotation rather than leaching was clearly the better method of handling the particular problem under consideration, before dismantling the experimental equipment, some data were secured on roasting and leaching the concentrate itself in competition with smelting. Some of the results obtained are given in the following paper and they are of particular interest at this time in view of the large quantities of flotation concentrates with their somewhat difficult smelting characteristics now being produced.

A complete wet process† consists of roasting and leaching the calcines in dilute sulphuric acid produced from the roaster gases, roasting the residue with salt and leaching with dilute tower liquors (the well-known Longmaid-Henderson process) and recovering the copper, silver and gold by cementation or electrolysis or a combination of both. It is evident, however, that the residue from the first leaching, carrying about 20 per cent. of the copper and all of the silver and gold, can be smelted if preferable. In considering the application of the scheme to individual cases, it must be remembered that freight plays a large part in any reduction process wherein smelting is not conducted at the mouth of the mine, and that it is not practicable today to build small smelting plants for individual operations.

The experiments may be grouped under four main heads: Roasting, leaching, chloridizing residue, and recovery of copper from solutions. The products of two concentrators were used: The Nacozari concentrates were the product of a large modern mill not using flotation, the copper mineral being largely chalcopyrite; and the Tyrone concentrates, the product of an experimental mill including flotation, the copper mineral being chiefly chalcocite. Typical analyses would be as follows:

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* Consulting Engineer.

† Patent applied for.

	Nacozari	Tyrone
Copper, per cent	14.0	14.0
Silver, ounces per ton	4.0	0.5
Gold, ounces per ton	0.01	Trace
Iron, per cent	31.0	28.0
Sulphur, per cent	34.0	30.0
Silica, per cent	13.0	
Alumina, per cent	3.0	
Lime, per cent	0.6	

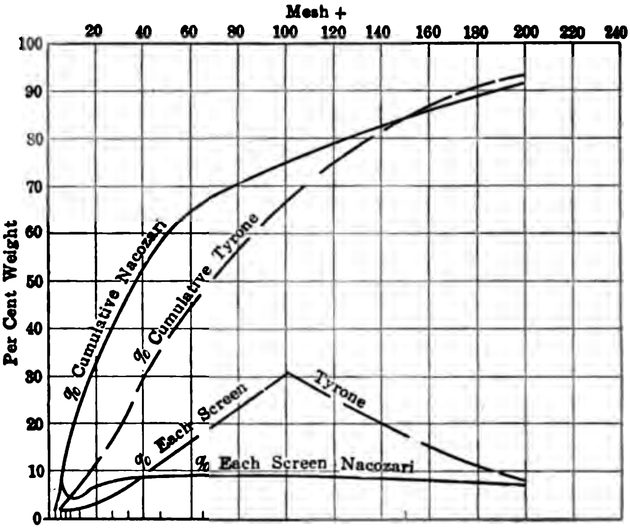

FIG. 1.—SCREEN ANALYSES OF CONCENTRATES CALCINES.

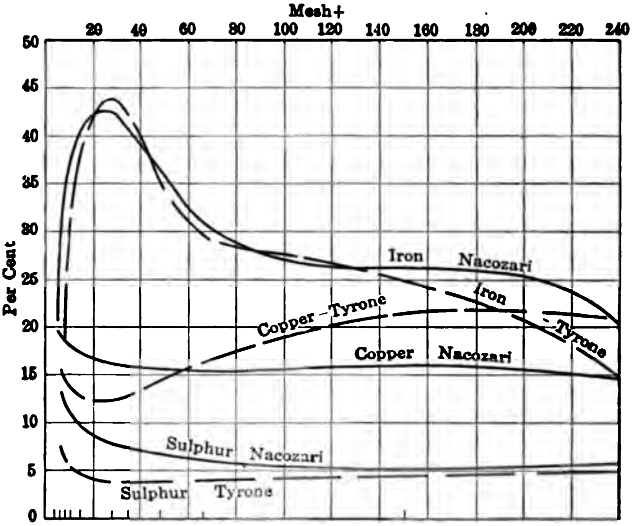

FIG. 2.—VARIATION OF COMPOSITION WITH SIZE OF PARTICLE.

Fig. 1 shows screen analyses of calcines obtained by dead roasting the two concentrates as described later. Fig. 2 shows the analysis of the calcines for copper, sulphur and iron for each screen size. The Nacozari concentrates carry considerable coarse jig product while the Tyrone material comes from an ore where the values are finely disseminated and the quantity of 100-mesh is quite marked. The presence of the flotation concentrates in the Tyrone material brings up the copper contents of the fine sizes.

Roasting

The object in roasting is to make as much of the copper and as little of the iron as possible soluble in dilute sulphuric acid. The work is similar to roasting pyrites fines in sulphuric acid manufacture, except that this solubility ratio rather than the complete utilization of sulphur is the controlling factor. Small-scale work is not very satisfactory as a guide to possible results as it is practically impossible to prevent overheating due to rapid oxidation of sulphur in a laboratory experiment. An 18-ft., water-cooled, six-hearth McDougall was used, the speed of rotation being gradually cut down until dead roasting conditions were obtained. Greater tonnages could doubtless have been obtained in a seven-hearth furnace.

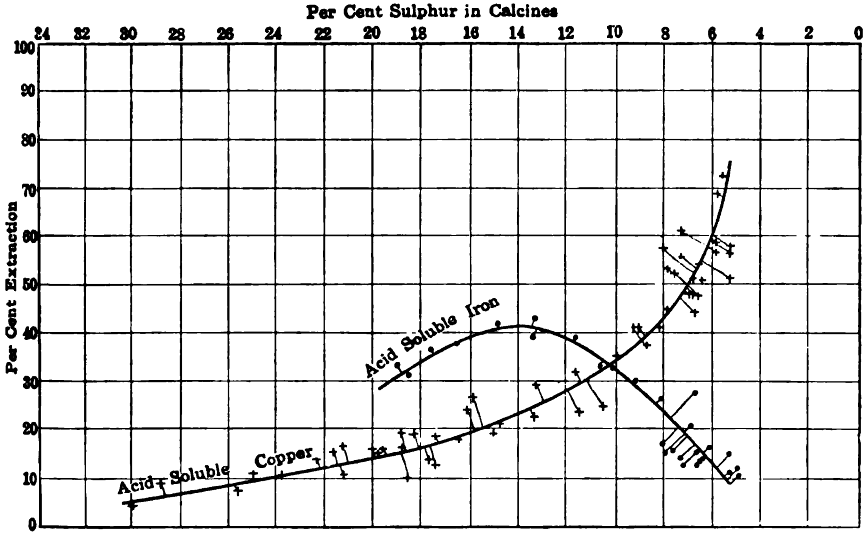

Fig. 3.—Nacozari Concentrates Roasted in 18-Ft. Six-Hearth McDougall Furnace and Leached in 4 Per Cent. Sulphuric Acid in Laboratory.

Many samples were taken from various hearths and the acid-soluble copper and iron determined by leaching with 4 per cent. H_2SO_4 in the laboratory. The results of these tests are given in Figs. 3 and 4. The results of tests on a series of sixth-hearth samples to determine the relation between tonnage and sulphur elimination are plotted in Fig. 5. It is

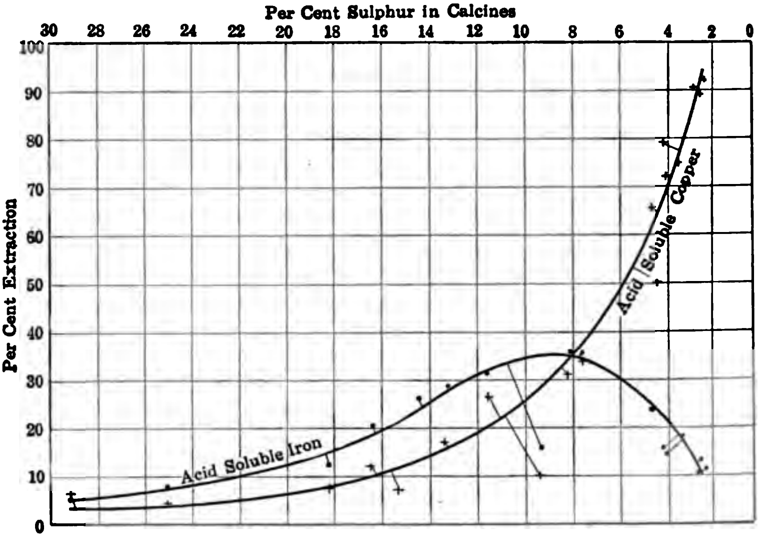

FIG. 4.—TYRONE CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

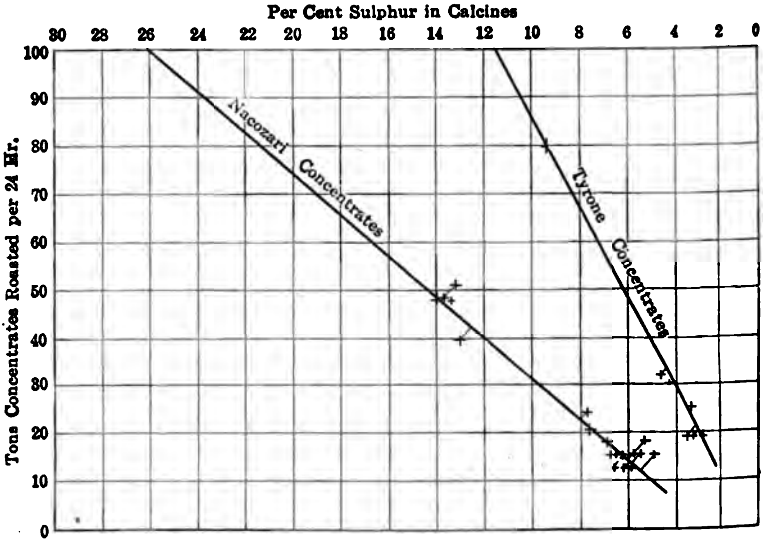

FIG. 5.—ROASTER CAPACITY VS. ELIMINATION OF SULPHUR.

evident that the chalcocite can be oxidized much more readily than the chalcopyrite, although size of particles has something to do with this. An investigation of the solubilities of the various sizes of particle was carried out by screening some of the calcines, as shown in Fig. 6. As would be expected, the finer particles are the more thoroughly oxidized; the jig product in the Nacozari concentrate is one reason for the poorer results obtained in the treatment of this material.

In general, these large-scale experiments indicate the possibility of reasonably obtaining the results desired—high copper and low iron solubility—but it is obvious that the residue after leaching will contain sufficient copper values to require retreatment, aside from the fact that any silver and gold will remain in this residue.

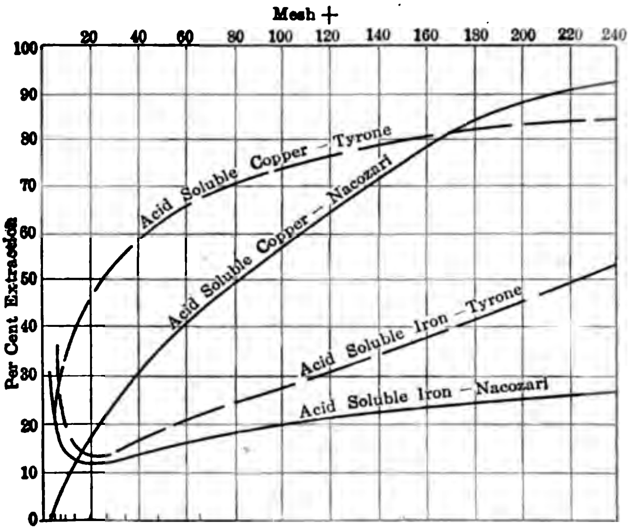

FIG. 6.—RELATIVE SOLUBILITIES OF VARIOUS SIZED PARTICLES OF CALCINE.

Leaching

As shown in the paper presented last year, such satisfactory results in the extraction of copper values from tailings were obtained by dumping the hot calcines from the furnace into a leaching trough, the few seconds' agitation thus obtained extracting almost as much as prolonged treatment in other apparatus, that the same idea was tried out with the concentrate calcines. It was not possible, for various reasons, to handle the output of the furnace directly, so the calcines were stored and then fed to a bucket elevator which in turn delivered into a V-trough in which the leaching liquor was flowing. The results were here disappointing, as although there was instant extraction of perhaps half of the soluble copper, a prolongation of the trough to give 60 sec. travel did not greatly increase this amount. It was definitely shown in the laboratory as well that prolonged agitation was necessary to extract all of the soluble copper. The leaching trough delivered into an acid-proof drag consisting of an endless belt with angle rakes, of the type commonly used in concentrators for dewatering. This acted more or less as a classifier, the very fine residues being carried over with the liquor, from which they were subsequently separated by settling. As this still gave insufficient agitation to the sands, a Parral tank was tried, but it was found that they were too heavy to yield readily to any sort of air-lift agitation. A Dorr classifier was then added to the apparatus and this did better. It was found, however, that it was necessary to pass the residues six or seven times through the leaching process in order to obtain an extraction equal to that shown by laboratory tests on the calcines. Fig. 7 gives a general view of the leaching apparatus.

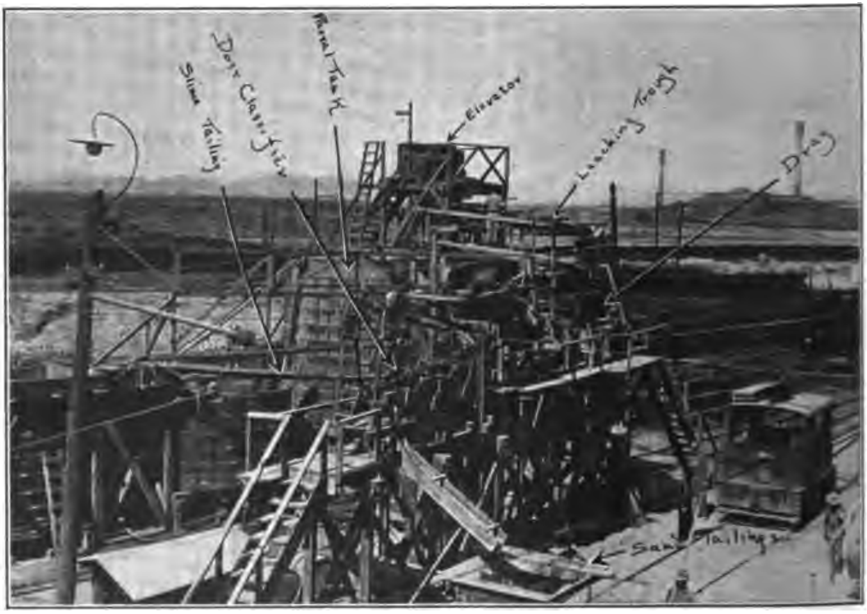

FIG. 7.—EXPERIMENTAL LEACHING PLANT.

The large-scale leaching tests were confined to the Tyrone material, a lot of 30 tons of calcines from some of the roasting tests being used. The first runs on a lot of 17 tons of not quite dead-roasted material running 4 per cent. sulphur gave results that were satisfactory except in that too much iron was dissolved, causing a needless consumption of acid and embarrassing any electrolytic scheme of recovery. Later, better-roasted material was available and a careful record kept of the metal balance and acid consumption, with the results shown in Table 1.

These figures check reasonably close by except in the case of iron; but it must be remembered that various iron parts in the apparatus used were

TABLE I.

Time Through	Per Cent. Cu in Tails	Trough Fed	Dorr Fed
1st	7.7	Acid	Water
2d	6.0	Acid	Water
3d	5.2	Acid	Water
4th	3.6	Acid	Acid
5th	3.3	Acid	Water

Extraction by Heads vs. Tails

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent.	Pounds	Per Cent.	Pounds
Heads	8,360	15.48	1,292	31.00	2,590	5.60	468
Tails	5,600	3.50	196	43.52	2,440	7.02	393
Extraction	.....	84.70	1,096	5.80	150	16.00	75
Extraction per lb. of Cu	.....	...	1.00	...	0.14	...	0.07

Extraction by Analysis of Liquors

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent.	Pounds	Per Cent.	Pounds
Heads	51,538	0.46	238	0.20	104	0.47	246
Tails	86,910	1.46	1,271	0.76	657	0.45	395
Extraction	......	80.00	1,033	21.30	553	31.90	149
Extraction per lb. of Cu	......	...	1.00	...	0.53	...	0.14

attacked by the liquor, which would artificially increase the iron taken into solution.

The acid consumption was 2,495 lb. of 100 per cent. H_2SO_4 for the run. This is equivalent to 2.28 lb. per pound of copper extracted. Laboratory tests on the same calcines indicated 2.0 lb. The leaching was done at about 125° F. with 5.6 per cent. free acid in the liquor entering the trough.

In general, when a 15 per cent. copper calcine is fed to the trough, the residue at the end of the trough will run about 8 per cent. Cu, the extraction representing the instantaneously soluble copper. This residue can be brought down to about 3.5 per cent. Cu by suitable agitating means, with a consumption of a little over 2 lb. of acid per pound of copper, and with the extraction of but little iron. The final residue weighs only about 60 per cent. of the original concentrate before roasting.

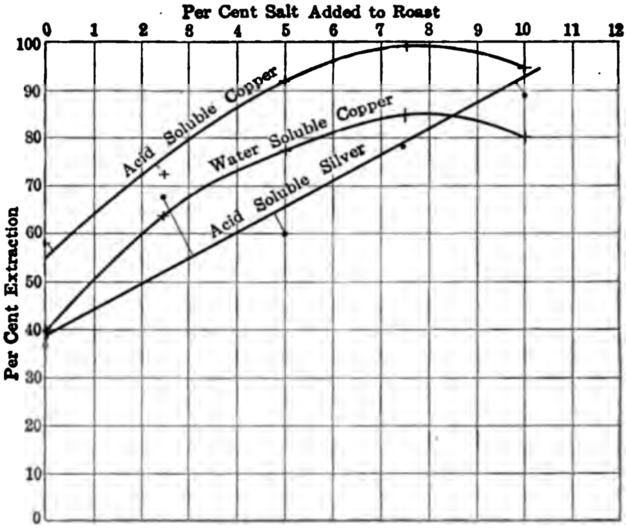

FIG. 8.—CHLORIDIZING LEACHED CONCENTRATES CALCINES.

Roasted 1½ Hr. at 975° F. with Addition of Salt and 10 Per Cent. Raw Concentrates. Calcines: 5.6 Per Cent. Copper, 1.9 Oz. Silver and 2.5 Per Cent. Sulphur. Raw Concentrates: 14.4 Per Cent. Copper, 0.55 Oz. Silver and 34 Per Cent. Sulphur. Liquor: 5 Per Cent. Na_2SO_4, 5 Per Cent. Na Cl, 5 Per Cent. Fe Cl_3 and 0.5 Per Cent. HCl + H_2SO_4.

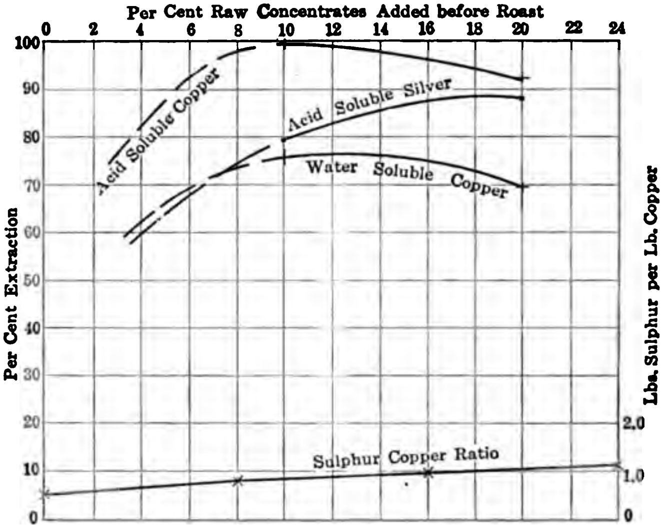

FIG. 9.—EFFECT OF SULPHUR-COPPER RATIO UPON EXTRACTION. 7½ PER CENT. SALT ADDED TO ROASTING "MIX".

Chloridizing Residue

No large-scale work was done on the chloridizing of the residues from the first leaching. The analysis of these residues, however, differs from that of pyrites cinder, so long successfully treated by this process, only in the amount of silica present. Various small-scale experiments were tried and 50 lb. or so were sent for test to a plant where the Longmaid-Henderson process was in operation. Both sets of experiments were entirely satisfactory.

A small lot of leached residues was prepared for test. These contained 5.6 per cent. Cu, 1.9 oz. Ag, and 2.5 per cent. S. Raw concentrates for adjusting the sulphur-copper ratio were used, containing 14.4 per cent. Cu, 0.55 oz. Ag, and 34 per cent. S. Fig. 8 shows the extractions with varying percentages of common salt added to the "mix" after roasting in an electric muffle furnace 1½ hr. at 975° F. and leaching in a liquor carrying 5 per cent. Na_2SO_4, 5 per cent. NaCl, 5 per cent. $FeCl_2$, and 0.5 per cent. HCl + H_2SO_4. Fig. 9 shows the effect of varying the sulphur ratio. The results show a 99 per cent. copper and a 79 per cent. silver extraction. The report on the lot of residues sent away fully confirmed these results.

Recovery of Copper from Solutions

The liquor from the chloridizing plant would doubtless be reduced to argentiferous copper cement by iron. But 20 per cent. of the original copper is involved. The sulphate liquor from the first leach could be precipitated on iron if desired, or with certain limitations would be suitable for electrolytic deposition of the copper and regeneration of the acid. Sulphide concentrates carry from 1 to 2 lb. of sulphur per pound of copper, equivalent to from 3 to 6 lb. of 100 per cent. H_2SO_4, less process losses, if the roaster gas is oxidized to sulphuric acid. Since the leaching calls for but a little over 2 lb. of acid per pound of copper, plus tailings losses, it would seem possible, therefore, to figure on a simple cementation plant, considering electrolysis as a competitor on a basis of relative profit and not of necessity.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Smelting at the Arizona Copper Co.'s Works

BY F. N. FLYNN,[*] CLIFTON, ARIZ.

(Arizona Meeting, September, 1916)

Introductory

In 1882, The Arizona Copper Co. Ltd., acquired producing copper mines at Metcalf and Morenci (locally called Longfellow). Metcalf is situated a distance of 7 miles, and Morenci a distance of 6 miles from the general office of the company at Clifton, Ariz. A reduction works with five blast furnaces was built in the center of the town of Clifton in 1884 and operated until January, 1914.

Concentration of oxidized ores was begun in 1890–1891, in a mill built adjoining the smelter, and in October, 1893, a leaching process was started for treating tailings from the oxide mill. A sulphide concentrator with 300 tons capacity was built at Clifton and started treating chalcocite ores in July, 1896. This was the first mill to treat low-grade copper-sulphide ores successfully. Another concentration mill for sulphide ores was erected at Morenci in 1900. The sulphide mills are now being remodeled to treat ores by concentration and flotation, with a capacity of 4,000 tons at Morenci and 500 tons at Clifton. The leaching plant has not been running since September, 1914.

The average yearly production from this company's mines and reduction works from 1885 to date has been:

	Pounds
10 years ended 1894	6,864,902
10 years ended 1904	20,439,614
10 years ended 1914	32,665,905
Part of year 1915, to Sept. 12	30,206,106
4 months of 1916, to June	15,964,840
June, 1916	4,900,000

Construction on a new smelting plant was started in February, 1912, and finished in February, 1914. In the *Transactions*, vol. 49, E. Horton Jones published "Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.," which fully describes the construction of this plant, with details of costs.

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[*] Superintendent of Smelting Department, The Arizona Copper Co., Ltd.

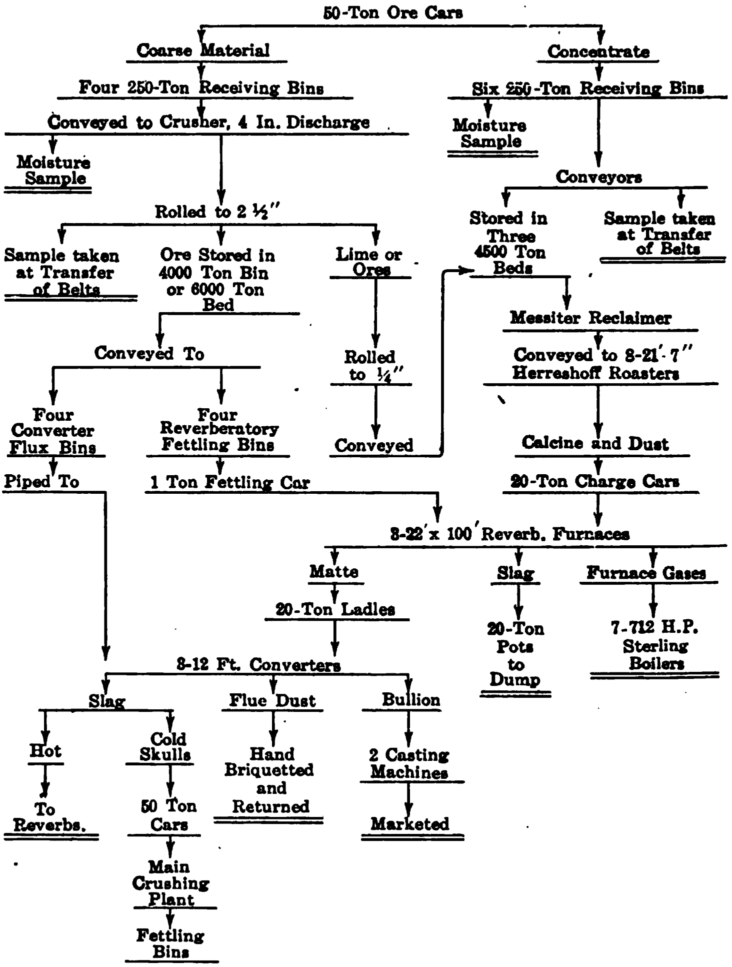
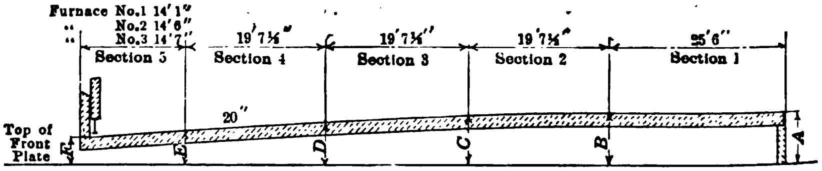


FIG. 1.—FLOW SHEET OF THE ARIZONA COPPER CO'S. SMELTING PLANT.



	Reverb. No.1	No.2	No.3		Reverb. No.1	No.2	No.3
A	7'8⅝"	6'10⅛"	7'10"	*D*	6'4½"	6'7¾"	6'0¾"
B	7'10⅜"	6'11½"	7'4"	*E*	5'4"	5'6⅜"	5'6"
C	7'0⅜"	6'9⅝"	6'10⅝"	*F*	4'0⅛"	5'0½"	4'8⅜"

FIG. 2.—SKETCH SHOWING POSITION OF REVERBERATORY ARCHES ON CENTER LINE.

The new smelter site is on gently sloping ground, 2 miles below the town of Clifton, and close to the San Francisco River. The Arizona & New Mexico standard-gage railroad and the Coronado narrow-gage railway serve the plant. Smelter yard standard-gage trackage consists of 2.38 miles for steam locomotives, and 1.72 miles for electric haulage.

Reference to the accompanying flow sheet will assist in following description of smelting practice at the new smelter.

Roaster Division

Roaster equipment consists of eight Herreshoff furnaces with steel shells 21 ft. 7 in. in outside diameter. Hearths and linings in the different furnaces are built of materials as shown in the accompanying table.

Hearth and Lining Materials in Herreshoff Roasting Furnaces

Furnace No.	Drier	Hearth Numbers From Top Downward						Shell
		1	2	3	4	5	6	
1	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
2	Red brick	Firebrick shapes	Slag-concrete	Firebrick shapes	Firebrick shapes	Firebrick shapes	2-in. concrete	Red brick
3	Red brick	Firebrick	Slag-concrete	Firebrick	Slag-concrete	Brick-concrete	2-in. concrete	Red brick
4	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
5	Red brick	Firebrick	Firebrick	Firebrick	Firebrick	Firebrick	2-in. concrete	Firebrick
6	Red brick	Firebrick	Firebrick	Firebrick	Slag-concrete	Slag-concrete	2-in. concrete	Red brick
7	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick
8	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick

Hearths designated "gravel concrete" were made of river gravel, 3-in. size and under, 2 parts; clean sand, 4 parts; and cement, 1 part. Slag-concrete mixture consisted of crushed slag (all the size that passed a ½-in. screen) 3 parts; clean sand, 3 parts; and cement, 1 part. Composition of this slag was: SiO_2, 47.0 per cent.; Al_2O_3, 8.5 per cent.; Fe, 27.5 per cent.; CaO, 3.3 per cent.; S, 0.5 per cent. In the fifth hearth of No. 3 furnace, crushed red brick was substituted for gravel in the concrete mixture.

The first nine concrete hearths built were reinforced with iron rods laid in top and bottom courses. Reinforcing in those with inner drop holes weighed 3,093 lb., and those with outer drops 2,148 lb. When fully heated the fourth and fifth hearths, with heavy reinforcement, rose 6 in. at the center of the arch, which necessitated chipping off some of the concrete from the center of hearths. In later construction ½-in. diameter reinforcing rods were used. This reduced the weight of iron to 1,172 lb. and 1,373 lb. respectively for inner and outer drop hearths.

Slag-concrete hearths are preferred, providing the slag is highly siliceous. All concrete hearths have given entire satisfaction. They have been in service since May 3, 1914, without any repairs.

Provision is made for cooling the central columns and rabble arms with air supplied from a motor-driven fan, but during the last 18 months cooling air has not been required.

The roaster charge has varied from ore, limerock and concentrate mixtures with 19.2 per cent. sulphur and 10.7 per cent. copper, to all concentrates with 30.7 per cent. sulphur and 15.8 per cent. copper, with fuel requirements varying from a maximum of 40 lb. of coal per ton of calcines to none when roasting all concentrate. The units of sulphur burned off have ranged from 7.5 to 17.4, the usual practice being to leave equal quantities of sulphur and copper in the calcine, because of the oxidized ore used in reverberatory fettling.

Normally, the material roasted requires only a little more heat than is derived from the sulphur burned. Heavy crude oil was first used as fuel, but with poor results, because with the low draft maintained burners could not be obtained of suitable size to burn continuously the small quantity of oil necessary to furnish this deficiency of heat, and distribute it properly over the hearth. The flame was admitted through one door on the third or fourth hearth from the top. A burner that would stay lighted in these comparatively cold hearths, produced an intense flame, and if kept running long enough to ignite the charge to the temperature desired, the brick hearth over the burner became too hot and charge fused on the hearth directly in front of the burners. Calcine produced was not uniform, because the sulphur had to be controlled by burning oil intermittently, which resulted in first a hot furnace and low sulphur, then a cold furnace with high sulphur.

Coal-fired Dutch-ovens with 12 sq. ft. of grate area have been adopted for supplying additional fuel required for roasting. Two ovens for each furnace admit the flame on the fifth hearth from the top. With these, the roast can be quickly and closely regulated, and the furnace repairs have been greatly reduced as compared with oil-burning conditions.

Ignition usually takes place on the fourth hearth from the top, and the temperature of the charge rises to 1,200 or 1,300° F. on the two lower hearths. With one reverberatory running, each roaster operated normally makes 65 tons of calcine per day. When two reverberatories are taking charge, each roaster has to supply 80 tons of calcine. Results show that while removing the same units of sulphur per ton, a furnace will consume less coal per ton when producing 60 than when producing 80 tons of calcine. Monthly averages of 113 tons of calcine produced per furnace day show a very high coal ratio. Furnace dampers are kept closed just to the point of smoking. Gases leaving the furnaces first enter a hopper-bottomed header-flue, through which they travel at a velocity of 6 or 7 ft. per second, at 325 to 425° F. temperature, and then pass into a large dust-chamber. Dust recovered from this chamber amounts to 0.8 per cent. of the dry charge to roasters.

The dust-chamber roof, made of No. 11 plate steel, shows slight deterioration at the laps. A sheet-copper cap laid over the joint at apex of roof, lasted 2 years and has been replaced by concrete.

Reverberatory Division

Smelting for production of matte is all done in reverberatories, either one or two furnaces being operated as required. Three furnaces set parallel to each other the long way, and 26 ft. apart, each with a hearth area 22 by 100 ft., are available, one of these always being a spare. The hearth in No. 3 furnace is crushed silica fused in on a slag bottom. Furnaces Nos. 1 and 2 have hearths of crushed silica fused in on bottoms of broken quartz. Roofs were all originally built with 1 in. to the foot rise in the arch. When the roof of No. 1 furnace was rebuilt, the arch was given a rise of 1¼ in. to the foot. Six-inch holes, spaced 24 in. centers, left in the roofs along the side walls and across the back, allow fettling material to be piped into the furnaces from trough-shaped bins built vertically over these holes. Five-inch diameter pipes on fettling bins are being changed to 8-in., to permit using coarser material without choking the pipe openings. Other dimensions of these reverberatories that affect their operation are given in an accompanying table.

Reverberatory Dimensions

	Furnace Number		
	1	2	3
Lowest matte tap hole to top of front plate	1 ft. 10⅝ in.	.....	1 ft. 3⅞ in.
Matte tap hole to skimming block, in use at present	10⅞ in.	.....	1 ft. 1 in.
Height of oil burners above front plate	1 ft. 11⅛ in.	.....	1 ft. 8⅜ in.
Height of oil burners above skim block at present in use	1 ft. 11⅛ in.	.....	1 ft. 5⅞ in.
Furnace throat area when slag line is level with top of front plate, square feet	17.4	26.2	22.0
Furnace throat area, when slag line is raised 6 in. above top of front plate, square feet	13.2	22.6	18.0
Area of opening in verb shaft, square feet	40.0	28.5	28.5
Area of intake opening to header flue, square feet	45.6	55.2	54.6
Thickness of side walls, as rebuilt, above matte line, inches	30.0	30.0	12.0

NOTE.—The 12-in. side wall is more favorable for greater height of fettling.

In furnaces Nos. 2 and 3 the weight of the shaft has been taken off of the verb arch, by means of a telescopic shaft supported on steelwork. Fig. 2 shows the relative positions of the reverberatory arches on the center line of roof. This height is satisfactory when burning small quantities of oil, but might be raised considerably in Sections No. 1 and 2 for larger quantities of oil.

The following log of the third campaign of No. 3 reverberatory furnace is representative of operations on a one-furnace schedule:

Furnace started May 20, 1914.
6.208 days lost in August, repairing side walls, verb shaft, and 24 ft. of new roof on Section 1.
0.769 days lost in September, October, November and December, 1914, making minor repairs.
1.122 days lost in January, 1915, patching roof and front end.
2.736 days lost in March, 1915. New roof on Section 1.
5.642 days lost in July, 1915. 50 ft. roof repairs on Sections 1 and 2, and new jambs under header-flue arch.
Furnace stopped Sept. 12, 1915, on account of labor strike, but was in good repair at that time.
16.477 days lost time in 16 months total campaign.

Dry Tons Solid Charge Smelted:

During total campaign	178,165.0
Daily average for total campaign	384.6
Daily average for one month, highest	512.4
Daily average for one month, lowest	312.2
Highest for one day, during highest average month	543.0
Lowest for one day, during highest average month	413.0

Barrels of Oil per Ton of Solid Charge:

Lowest month	0.653
Highest month	0.951
Average during campaign	0.744

By a campaign is meant the elapsed time between starting a furnace and tapping out the matte. New roofs on Sections 1 and 2 were built upon ore centers, without tapping the matte. The last side walls repaired were in November, 1914. Since adopting greater fettling height, no further wall repairs are anticipated.

Two parallel charge tracks cross the furnaces near the back in a direction at right angles to their length. Fifteen-ton charges of calcine, at a temperature of 1,050 to 1,150° F., are dropped through one water-jacketed charge hole in the center of the roof on either track, but 90 per cent. of all charges are dropped from the back track. This part of the charge dropped through the roof is designated "direct charge." Seventy-five per cent. of the total solid charge smelted is introduced in this way.

Bad furnace conditions are always expected if such cold material as crude ore, slag or limerock of any size, or anything coarse like siliceous ore agglomerated with converter slag, either cold or warm, is dropped as direct charge without mixing it well with hot calcine. Ore, limerock or clinker produces "charge floaters." Cold slag sinks and makes sticky furnace bottoms. When it is necessary to add cold or coarse material to the direct charge, the best results are obtained when they are well mixed with hot calcines, in the calcine charge cars, but better still in the roasting furnaces.

Bed mixtures to be roasted are made of the required composition to produce a slag, when smelted, in which the ratio of oxygen in acids to oxygen in bases will fall between 2.00 and 2.25. A vast difference, however, is found in the smelting properties of calcines made from these mixtures, regardless of the similarity between their analyses or the analyses of the slags they produce. Calcine made from concentrate or sulphide ores of ½ in. size smelts readily, whereas if the calcine is an artificial mixture made of siliceous oxidized ore crushed to this size, and mixed with an excessive proportion of limerock, the charges melt more slowly. The results from the two following mixtures will fairly illustrate this point:

Class of Material	Mix 103. Per Cent. of Total	Mix 120. Per Cent. of Total
Sulphide concentrate	100.0	70.7
Siliceous ore, low sulphur	.....	4.8
Siliceous oxide ore	.....	12.9
Limerock	.....	11.6
Total	100.0	100.0

Mix No.	Analysis						
	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Cu, Per Cent.	Oxygen Ratio
103	18.3	4.6	28.2	0.7	29.0	14.10	1.43
120	20.7	4.7	24.1	7.6	22.5	10.98	1.45

A daily average of 365 tons of calcine per furnace was smelted during the time that mix 103 was on the charge. In comparison, 200 tons per furnace day was barely maintained while the charge consisted of calcine from mix 120. The practice is to crush siliceous ore as fine as 3/16-in. for roaster beds, and endeavor to keep the sulphide material as high as

75 per cent. of the total bed mixture. Ore and limerock mixtures require a much higher oil ratio than calcine from straight concentrate.

In other words, the greatest percentage of oxidized material which it is permissible to smelt in the reverberatory furnace is found to be:

	Tons per Furnace Day	Per Cent.	
Direct charge	300	25	Oxidized ore and limerock.
Fettling	100	100	Oxidized ore.
	—	—	
Total	400	44	Oxidized ore equivalent.

Above 44 per cent. oxidized ore, the charge could undoubtedly be better handled in a blast furnace. The charges spread better when the furnace is nearly full of matte, and also when the copper and sulphur in the calcine are equal. Matte under 40 per cent. copper is favorable for higher tonnage.

Fettling.—Ore and byproducts crushed to a 2½-in. size for fettling constitute the remaining 25 per cent. of the solid charge smelted. It is customary after each skim to feed these materials from bridge-wall to shaft in the following proportions:

Furnace	Section No.	Per Cent.	Material
Back	1	25	Old smelter slag. Converter slag skulls.
Back	2	50	Less siliceous ore.
Middle	3	12	Less siliceous ore.
Middle	4	8	Less siliceous ore.
Front	5	4	More siliceous ore.
Shaft	..	1	Silica.

	Analysis							Oxygen Ratio
	SiO_2, Per Cent.	Al_2O_3 Per Cent.	Fe, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	S, Per Cent.	Cu, Per Cent.	
Converter slag skulls	19.6	3.9	47.2	0.9	0.0	3.6	9.00	0.89
Old smelter slag	37.2	8.9	22.0	10.3	5.2	0.3	3.56	2.12
Less siliceous ore	42.6	7.7	18.0	3.0	2.5	1.0	6.80	3.74
More siliceous ore	65.0	12.2	5.6	0.8	0.6	1.9	3.78	19.43
Silica	84.4	6.3	3.1	0.5	0.0	2.6	0.74	46.34
Approximate average	40.4	7.5	21.5	3.5	2.4	1.3	6.49	3.08

Prior to July, 1914, the height of fettling was carried not to exceed 6 in. above the slag line. This height was gradually increased until October, 1914, when the practice of fettling to the holes in the roof was commenced and later successfully adopted after the size of fettling material was increased from ¾ in. to the present 2½-in. size. The coarse size stands better. It reposes at an angle of 45° above and 45° to 60° below the slag line.

The results achieved in performing fettling operations as practised here, depend upon the relative location and manner in which the material is dropped, as well as the amount of fines in it and its chemical and physical composition, as may be concluded from the following account of experience with different fettling materials.

1. Slag and byproducts crushed fine work unsatisfactorily in any part of the furnace. However, if crushed coarse, results are good when such material is charged in the back of the furnace.

2. Less siliceous fine ore is not good fettling in any place in the furnace. It does not stand, and requires care to keep it from running out into the furnace. Crushed coarse, it makes good fettling wherever used.

3. More siliceous fine ore mixed with coarse byproduct make a bad combination. The only place the more siliceous fine ore alone can be used with any degree of success is on the bridge-wall.

4. More siliceous coarse ore is not so acceptable as less siliceous coarse ore, except in the front half of the furnace, because it makes too much siliceous scum on top of the slag.

5. Coarse raw concentrate is good in the back half of the furnace, but does not stand so well nor so high as coarser material. It gives bad results in the front half because it melts easily to a magnetic mush.

6. Fettling with hot calcine can only be done with any degree of success in Section No. 1, and then only slightly above the slag line. At that elevation it runs out over the furnace bottom. An attempt was made to fettle the entire furnace with calcine, but was abandoned when the tonnage dropped to less than half capacity. Calcine gave poorer results than any other material experimented with.

7. Pyrite in the front half melts easily to a magnetic mush. In the back half when mixed with siliceous oxide ore it works well.

8. The ideal fettling material was found to be less siliceous copper-sulphide ore containing 5 to 10 per cent. sulphur.

In other words, basic or sulphide materials are charged near the firing end because of their lower fusing points. If charged in the front portion, where the temperature is lower, they melt with sufficient ease to make a thick, mushy, semifused slag, and in the case of pyrite make quantities of magnetite. Siliceous ores with 60 per cent. silica produce a scum which blankets the slag and causes higher oil ratio and lower tonnage. If the siliceous ores contain much copper in oxidized form, the scum will assay from 0.7 to 1.3 per cent. copper, whereas if the copper is present as a sulphide the scum will assay very much lower.

The most important development as regards fettling has been along the line of coarser crushing and greater height of material to protect the side walls without encroaching on the furnace bottom area. Fettling "floaters" are unknown. The successful results of this practice are best seen from the photograph of one of the furnaces (Fig. 3).

Fuel is 14 Bé., 18,000 B.t.u., California crude oil, pumped at 120 lb. pressure and delivered through steam heaters at a temperature of 170° F. to six burners and atomized with converter air. Two hundred and forty to 365 bbl. of oil is burned per furnace-day. This variation covers a one-furnace schedule when hard firing for high tonnage is required, or two furnaces with light firing and correspondingly low tonnage smelted.

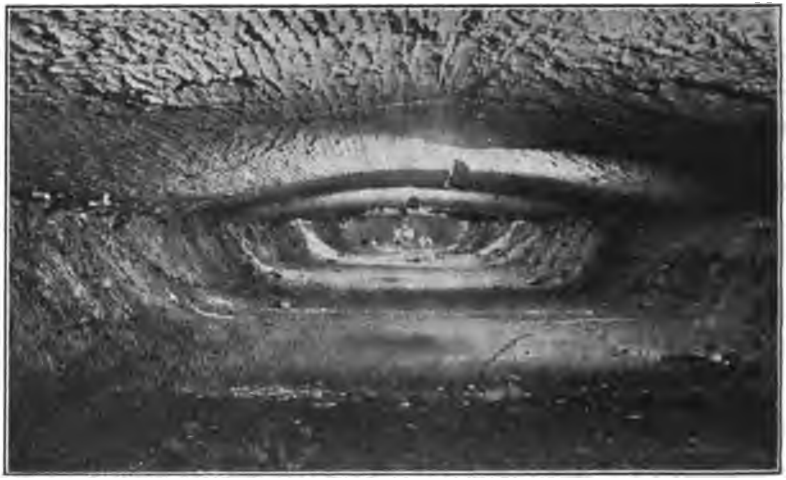


FIG. 3.—REVERBERATORY FURNACE NO. 3 OF ARIZONA COPPER CO., LTD.

Campaign No. 3. Started May 20, 1914, finished Sept. 12, 1915 (date of labor strike.)

Dry tons of solid charge smelted: total 178,165; highest daily average for one month, 512.4.

Fettled to the roof with "less siliceous" oxide ore and converter slag skulls, crushed to 1½ in. Slope of fettling, from 45 to 60°. Fettling material amounts to 25 per cent. of total solid charge.

Furnace-throat draft (inside the furnace) is 0.12 to 0.16 in. of water, and in header-flue common to all furnaces, draft is 0.50 in. water. Gases leave the furnaces at temperatures from 1,800 to 2,000° F. Seven Stirling boilers are available for reverberatory waste heat, each 712-hp. capacity, and all connected in multiple through cross-over flues from the header-flue. Dampers are so arranged that gases can be sent to any one or all of the boilers. There is no bypass to the chimney. Four or five boilers are used when running one furnace. Reverberatory dust caught in all the boilers does not exceed 5 tons monthly when running one reverberatory.

Furnace gases leaving the waste-heat boilers at temperatures varying between 471° and 588° F., are conducted through 6-ft. diameter steel pipes into a flue connecting with the main chimney. The temperature of gases in the flue ranges between 400° and 500° F. at different points, and at these temperatures the average velocity of gases is 10 ft. per second, when running one furnace. An average analysis of this gas is SO_2, 0.3 per cent.; CO_2, 5.9 per cent.; O, 12.9 per cent.; CO, 0.4 per cent.; N, 80.5 per cent.; SO_3, not determined. When the flue was cleaned recently, 100 tons of dust was removed that had accumulated during 2 years' furnace operations. The composition of this dust was:

	SiO_2	Al_2O_3	Total Fe	Ferrous Fe	CaO	S	Cu	Free H_2SO_4	H_2O
Reverberatory chamber dust	17.4	6.3	11.8	....	2.3	10.3	9.65	3.0	9.5
Soluble in water	....	1.8	3.7	2.6	2.1	9.1	6.60		
Per cent. of total, soluble in water	....	28.3	31.4	....	91.4	88.3	68.4		

The flue roof was originally made of No. 14 steel plate. This had to be repaired after 8 months' service. Holes were eaten in the plate and in spots the whole sheet was pitted and nearly eaten through. Moist flue dust covered the inside surface of the roof. This deposit consisted principally of sulphates of iron and copper, as is shown by the following composition calculated from analysis of dry sample:

Insoluble	3.0
$CuSO_4$	7.7
$Al_2(SO_4)_3$	8.0
$FeSO_4$	18.7
$Fe_2(SO_4)_3$	45.3
$ZnSO_4$	3.2
$CaSO_4$	2.9
Free H_2SO_4	12.3
Total	101.1

This would not jar loose from the sheet, and metallic copper had deposited on the iron. Concrete slabs reinforced with wire netting were laid on top of the steel plate and the joints between slabs filled with asphalt. The concrete covering has served 2 years. The corrosive action of gases on the iron was arrested for 10 months of operations after the slab roof was put on, but during an idleness of 5 months, due to labor trouble, the plate had corroded completely.

Lime in the mortar used in laying tile walls of this flue was partly changed to sulphate by the furnace gases. While the flue was cold, the walls swelled far out of plumb. The tile broke and walls had to be rebuilt. New walls are brick laid in slime mortar.

Converter slag is poured through the back wall of furnace, at the center, through a launder discharging at the height of the oil burners. Magnetite builds up under the discharge of the launder, but the bridge-wall on either side of it requires fettling.

Considerable has been written regarding the fluxing value of converter slag poured into reverberatory furnaces. The prevailing opinion seems to be that very little fluxing value is within one's control. It is argued that the basic converter slag does not act upon the charge resting on the slag surface, does not mix with the trisilicate slag, and passes through the furnace like a submarine as a monosilicate, emerging near the skimming end. The technical reason advanced for these arguments is the relatively high specific gravity of the basic converter slag. The writer has proved to his own satisfaction that practically all of the iron present in liquid converter slag, other than iron in the magnetic state, is effective as flux. The violent boiling and the reactions that take place when calcine charge is dropped through the central charge hopper are effective in mixing any slag that may have separated into layers in the furnace at the back. If it is true that the charge dropped in the back of the furnace partly melts to a trisilicate slag, underlaid by the bisilicate, which in turn rests upon the monosilicate converter slag, it would be a difficult task to explain why the surface or trisilicate slag is so active in its consumption of fettling material at the slag line. On the other hand, it is not difficult to imagine the monosilicate slag attacking the fettling material along the side walls, from the matte line up. How it would fail to do so would be difficult to explain.

The effect of liquid converter slag was vividly shown on one occasion, when a furnace had been tamped with moist quartz from floor to roof along the walls, and when fused was largely filled with hot converter slag and little regular charge. Within 24 hours there was little quartz remaining.

On several occasions, two furnaces running on identical charges have been used for illustrative purposes. Into one, all of the converter slag was poured. In the other, without converter slag, the full quantity of limerock equivalent to the total fluxing value of the converter slag was required.

There are others who contend that even magnetite in converter slag is acted upon by the furnace matte and becomes effective as flux. Our own experience is that whenever magnetite accumulates in the furnace, it must be pulled out with the rabble as mush, the necessary limerock added to the charge, and plenty of sharp drills made ready at the matte tap hole.

The absence of an accurate method for the determination of magnetic oxide of iron in furnace byproducts is one of the most unfortunate and regrettable conditions with which the metallurgist is confronted, and renders the elucidation of this obscure subject difficult. Without some means of quantitatively determining this substance, accurate data as to its fluxing value can not be obtained from balance sheets even if materials and products are all weighed, because it is not certain from an analysis that the magnetic iron has been changed in passing through the furnace.

Converter Division

The production of magnetite in the converters is, within reasonable limits, controlled by the converting practice. The converter slag poured into the reverberatory furnace should not contain any magnetite visible to the eye. The safest practice is to reject the last 2 cu. ft. of slag when pouring. The rejected slag with the ladle skull will eventually enter the furnace as fettling material, where its detrimental effect to the furnace is temporarily turned to some good, because of its refractory properties. Its tendency to stick to the sides when mixed with cold ore is quite noticeable when the fettling bank is removed from the walls when the furnace is out of service. By this practice, magnetite is made to perform a somewhat similar service in the reverberatory furnace to that which it performs in the protective coating in a basic converter. Unfortunately only a small portion of the total magnetite production can be so used.

Some metallurgists prefer to make a small quantity of magnetite on each converter charge, contending that in that way the coating remains a more constant thickness. They accomplish this by blowing a few minutes on the start or finish, either or both, but usually at the start, with a shortage of silica. The writer is of the opinion that by intentionally making magnetite with every charge; the quantity made must be very large as compared with the small amount which can be made to stick to the walls by so haphazard a practice going on continuously. Our practice is to coat the shells when the lines of brickwork can be seen through the thin coating. And, when a shell is being coated, no attempt is made to do anything with the matte other than form the coating. When coated, the magnetite and copper mush is dumped on the floor under the shell; later it is charged in small quantities at intervals in other charges. When coated, a shell is never blown without a large excess of silica. Although siliceous ore is used for flux, 350 lb. of 80 per cent. silica is used to granulate the final slag on the copper finish, to insure the removal of the last traces of iron from the copper with the more active silica. Since this quartz, with any magnetite which it has collected, remains in the converter for the next charge, the magnetite is in contact with the quartz particles from the start of the blow.

The converting division is equipped with five 12-ft. Great Falls-type shells lined with magnesite brick; three converter stands, each operated by an independent motor (the fourth stand is under construction); two 40-ton cranes; two straight-line casting machines, each with 39 copper molds. Two stands are regularly used, for which air at 13 lb. pressure is supplied from power house by two of three Nordberg blowing engines.

When converting matte of over 40 per cent. copper, only a small amount of cold matte shells and cleanings from the floor can be used; but with lower-grade matte, all matte shells, converter flue dust, and any converter byproducts made except converter-slag skulls can be handled in this division.

Molten copper is transferred from converters to casting machines in cast-steel ladles lined with fines screened from ores regularly used as converter fluxes. Bars are cast in molds made of converter copper. A 1¼-in. cast-iron splash plate covers half of the bottom area, and an average of 73 tons of bullion is cast per mold. Bars weigh 240 lb. each and 35 min. is required for casting a charge weighing 7 tons. Considerable chipping of bullion bars is necessary to remove edges and fine shot due to blowing to gas finish of 99.60 per cent. copper.

Comparison of One-Month Periods with One and Two Reverberatory Furnaces in Operation

Roaster Division	One Reverberatory	Two Reverberatories
Total tons dry charge roasted	14,257.0	17,235.0
Dry tons charge per furnace-day running time	78.4	73.9
Analysis of charge:		
SiO_2, per cent.	17.3	19.8
Al_2O_3, per cent.	4.7	4.5
Fe, per cent.	28.2	24.3
CaO, per cent.	1.6	6.4
S, per cent.	28.8	24.5
Cu, per cent.	13.61	12.08
Oxygen ratio	1.33	1.44
Analysis of calcine:		
SiO_2, per cent.	20.0	22.0
S, per cent.	13.8	13.8
Cu, per cent.	15.70	13.44
Weight of calcine produced as per cent. of dry charge	86.7	89.9
Units sulphur eliminated	15.0	10.7
Pounds coal used per ton of roaster charge	4.2	14.2
Waste-gas temperature	428.0	396.0

Reverberatory Division	One Reverberatory	Two Reverberatories
Dry tons direct charge per furnace-day	402.5	262.7
Dry tons fettling per furnace-day (ores)	84.9	65.7
Dry tons fettling per furnace-day (byproduct)	25.0	38.7
Dry tons total solid charge smelted per furnace-day	512.4	367.1
Per cent. of fettling to total solid charge	21.45	28.3
Analysis of direct charge:		
SiO_2, per cent	19.8	21.7
Al_2O_3, per cent	5.4	4.9
Fe, per cent	32.3	26.6
CaO, per cent	2.3	7.7
S, per cent	13.7	13.6
Cu, per cent	15.55	13.24
Oxygen ratio	1.32	1.41
Analysis of fettling (ores and byproducts):		
SiO_2, per cent	38.8	39.5
Al_2O_3, per cent	7.6	8.0
Fe, per cent	23.4	25.5
CaO, per cent	1.7	4.0
S, per cent	9.1	3.0
Cu, per cent	6.65	5.52
Oxygen ratio	3.36	2.93
Total tons hot converter slag to reverberatories	2,419.0	2,862.0
Analysis of reverberatory slag:		
SiO_2, per cent	35.6	38.0
Al_2O_3, per cent	9.7	9.0
FeO, per cent	49.7	39.4
CaO, per cent	2.4	8.4
MgO, per cent	0.6	1.5
Cu, per cent	0.51	0.45
Oxygen ratio	2.04	2.06
Analysis of matte, per cent. Cu	37.43	37.70
Per cent. matte fall	31.68	27.1
Per cent. sulphur volatilized in furnace	24.14	25.7
Bbl. oil burned per ton solid charge smelted	0.653	0.778
Bbl. oil burned chargeable to steam	0.315	0.349
Bbl. oil burned account smelting	0.338	0.429

The accompanying 2 months' results were selected for comparison, because they show the extreme variation in the material treated in this department. Limited storage capacity at the smelter required two furnaces to be operated a part of the time to handle receipts. Additional charge needed for the month that two reverberatories were operated was composed of siliceous oxidized ores and limerock, and represents a temporary condition. The concentrator additions, now almost finished, will in the future supply sufficient concentrates to make the proportions of total material treated fall somewhere within the percentage limits given in the examples.

Converter Division	One Reverberatory	Two Reverberatories
Tons reverberatory net matte treated	5,293.0	5,903.0
Per cent. Cu in reverberatory net matte treated	37.43	37.70
Tons siliceous ore used for flux	820.0	1,356.0
Analysis of siliceous ore used:		
SiO_2, per cent.	64.5	65.3
Al_2O_3, per cent.	10.6	11.6
Fe, per cent.	5.2	5.7
CaO, per cent.	0.2	0.5
S, per cent.	0.5	1.7
Cu, per cent.	6.45	4.29
Oxygen ratio	25.30	22.65
Analysis of converter slag produced, per cent. SiO_2	19.6	19.6
Tons copper bullion produced	1,800.0	2,450.0
Cu in bullion, per cent.	99.60	99.55
Tons copper produced per converter-day running time	43.9	51.4
Minutes blowing per ton copper	33.0	28.0
Average tons bullion per charge	6.429	8.524
Average time of blowing a charge	3 hr. 54 min.	3 hr. 59 min.
Tons cold new material treated per ton bullion	0.462	0.592
Tons cold byproduct re-treated at converters per ton bullion	0.849	0.577
Blast, pounds pressure	13.0	13.0
Cu. ft. air per converter per minute	6,738.0	5,445.0
Cu. ft. air per ton bullion produced	220,949.0	152,499.0
Cu. ft. air per ton iron and sulphur eliminated	135,043.0	96,145.0
Oxygen efficiency, calculated from blower displacement and requirements of Fe and S eliminated	60.63 per cent.	84.09 per cent.
Total Material Treated—All Divisions:		
Dry tons concentrates	13,227.0	13,360.0
Dry tons ores	4,069.0	7,303.0
Dry tons limerock	391.0	2,130.0
Dry tons new material, total	17,687.0	22,793.0
Dry tons byproducts treated at reverberatories	924.0	*2,317.0
Concentrates, per cent. of total new material treated	74.78	58.62
Ores, per cent. of total new material treated	23.01	32.04
Limerock, per cent. of total new material treated	2.21	9.34

* Including reverb. slag skulls.

Boiler Plant

There are 10 Stirling boilers, three 384-hp. direct-fired, and seven 712-hp. waste-heat. Water is measured in total and also separately to the two sets of boilers. Oil is measured to the direct-fired boilers.

All boilers contain Foster superheaters, and deliver steam to the mains at 175 lb. pressure. The temperature of steam delivered at the power house is about 475° F. The temperature of the waste gases leaving the boilers varies between 471° to 588° F., depending upon the furnace practice and the number of boilers in service. The gases from one furnace practice are passed through five boilers. When two furnaces are in commission, six or seven boilers are used.

When smelting in one furnace, three direct-fired boilers are in continuous service to supply the power requirements, but the night load is light, therefore the boiler efficiency is low. The pounds of water evaporated per pound of oil averaged 11.98 over a period of 8 months. The oil averages around 15° Bé. and 18,200 B.t.u.

With two furnaces in operation, there is sufficient waste heat to generate all steam requirements for the night load, but oil must be used to carry the day load. Under these conditions, rather than keep the oil-fired boilers under steam over night, the three oil-fired boilers are not used. Instead, oil is burned under the waste-heat boilers. The oil so burned is measured and credited with evaporating 11.98 lb. of water, in the same manner as under oil-fired boilers on a one-furnace basis.

The net steam delivered to the power-house engines is credited to the boiler plant. Steam used in heating and atomizing oil and for other miscellaneous purposes, as well as steam wasted, is absorbed in the furnace-oil costs.

Under one-furnace conditions, the waste-heat boilers averaged over a period of 8 months, 7.71 lb. of water evaporated per pound of oil burned in the furnace.

The furnace-boiler division receives credit from the power house varying from 36 to 56 per cent. of the oil burned in the smelting furnace.

The indicated horsepower from the waste-heat boilers, per ton of solid charge smelted on a one-furnace basis over a period of 9 months averaged 98.71 i.hp.-hr. whereas on a two-furnace basis, over a period of 7 months, the average was 114.34 i.hp.-hr. from waste heat only.

Smelter Power

Divisions	Kilowatt-hours		Kilowatt-hours per Ton New Material	
	One Furnace	Two Furnaces	One Furnace	Two Furnaces
Sampling:				
Receiving, high-pressure air converted to kw.-hr	1,813	2,590	0.10	0.11
Crushing, electric	10,501	21,548	0.59	0.95
Sampling, electric	1,146	2,351	0.06	0.10
Bedding, electric	3,819	7,836	0.22	0.34
Reclaiming, electric	22,576	22,843	1.28	1.00
Total	39,855	57,168	2.25	2.50
Roasting:				
Roaster furnaces, electric	7,692	7,495	0.43	0.33
Calcine cars, electric	3,140	5,093	0.18	0.22
Total	10,832	12,588	0.61	0.55
Reverberatories:				
Reverb. furnaces, electric	3,013	3,303	0.17	0.14
Reverb. furnaces, high-pressure air converted to kw.-hr	453	647	0.03	0.03
Reverb. furnaces, low-pressure air converted to kw.-hr	37,339	72,774	2.11	3.19
Slag railway, electric	4,420	10,451	0.25	0.46
Boilers, high-pressure air converted to kw.-hr	1,813	2,590	0.10	0.11
Boiler feed-water pumps, electric	31,220	20,553	1.77	0.90
Oil pumps, electric	4,030	2,009	0.23	0.09
Total	82,288	112,327	4.66	4.92
Converting:				
Converters, electric	5,263	5,674	0.30	0.25
Converters, low-pressure air converted to kw.-hr	218,235	206,802	12.34	9.07
Cranes, electric	22,790	19,635	1.29	0.86
Bullion casting machines, electric	2,340	2,920	0.13	0.13
Total	248,628	235,031	14.06	10.31
General Works:				
River well, electric	6,760	8,001	0.38	0.35
Boiler and machine shops, electric	2,900	2,730	0.16	0.12
Boiler and machine shops, high-pressure air converted to kw.-hr	4,986	7,122	0.28	0.31
Yards, electric	1,313	1,439	0.07	0.06
Laboratory, electric	2,807	3,077	0.16	0.13
Total	18,766	22,369	1.05	0.97
Total, all divisions	400,369	439,483	22.63	19.25
Dry tons new material treated	17,687	22,793		

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Leaching Tests at New Cornelia

BY H. W. MORSE, PH. D., LOS ANGELES, CAL.,* AND H. A. TOBELMANN,† AJO, ARIZONA

(Arizona Meeting, September, 1916)

INTRODUCTION

THE experimental work on the oxidized copper ore at the New Cornelia mine at Ajo, Ariz., ended on Jan. 12, 1916. On that date final decision was made on the general nature of the process to be used in the 5,000-ton leaching plant, and on many of the details, as far as experience on a40-ton scale could decide them.

With the approval of the Board of Directors and the General Manager, John C. Greenway, we have compiled what seem to be the most interesting data on the results obtained during the experimental period. The most important part of the data resulted from the operation of a 1-ton and a 40-ton plant at Ajo.

Experimental work on Cornelia oxidized ore dates back to April, 1912, and has been going on nearly continuously since that time. A good many variations from the original idea have been tried, but the process finally decided upon is in principle and in all of its details a simple one. The history of the leaching work on this ore has been brought down to about a year ago in papers presented to this Institute by Stuart Croasdale[1] and Dr. L. D. Ricketts.[2] The present paper will, therefore, deal principally with the results obtained during the last year of the work.

PRELIMINARY TESTS BY MR. CROASDALE

Preliminary tests were begun by Stuart Croasdale in July, 1912. A number of important points were definitely decided by his work. Among these were:

1. New Cornelia oxidized ore, not very finely crushed (to about 2- to 3-mesh) showed good extraction with 5 per cent. sulphuric acid. It appeared that better than 80 per cent. extraction could be expected.
2. The consumption of acid for this extraction was not prohibitive. It was apparently 3.5 to 4 lb. gross per pound of copper recovered.

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[1] Leaching Experiments on the Ajo Ores, *Trans.*, vol. 49, p. 610.

[2] Some Problems in Copper Leaching, *Trans.*, vol. 52, p. 737.

3. It was proposed to make cement copper. The iron consumption per pound of copper recovered was a little over 1 lb.

4. A column 12 ft. deep of ore crushed to this fineness could be successfully percolated.

During Mr. Croasdale's work the plan in view was a very simple one indeed. The ore was to be leached with sulphuric acid. Cement copper was to be produced by the use of metallic iron. As Dr. Ricketts explained to the Institute nearly 2 years ago, the oxidized Cornelia ore could apparently be leached and cement copper produced at a profit, if sulphuric acid and scrap iron were bought in the open market and shipped to Ajo, and cement copper shipped out, all solutions being thrown away after removal of the copper.

Mr. Croasdale showed also that the amount of substances other than copper, taken into solution under his conditions of crushing and leaching, is small. This naturally suggested the possibility of a closed leaching cycle with electrolytic deposition and at least partial regeneration of the acid required for solution of the copper. Early experiments showed that iron, which is dissolved from the ore to a certain extent, and oxidized at the anode during electrolysis, was a troublesome factor.

There seemed to be three possible ways of meeting this difficulty: (1) The use of a diaphragm; (2) the use of a depolarizer; (3) purification from iron before electrolysis.

ANTISELL PROCESS

About this time there came to the notice of the manager good results that were being obtained with a special anode, the invention of F. L. Antisell. This anode was in the form of a long, deep, narrow box with wood veneer sides, containing graphite electrodes and having the space between graphite and wood filled with selected coke.

A cyclic process was tried with these anodes. Ore crushed to 4-mesh was leached 3 days by upward percolation, with a practically constant solution containing 3 per cent. H_2SO_4, 2 per cent. ferrous iron, 0.5 per cent. ferric iron and 2 per cent. Al_2O_3. Sulphur dioxide gas was used to keep down the ferric iron concentration.

The process gave good results. The anode was, however, rather cumbersome and the absorption of the sulphur dioxide was not satisfactory, presumably owing to the rather high acid content of the solution.

The following is a summary of average results obtained by the Antisell Process: Tons of ore leached, 75; heads, 1.68 per cent. Cu; extraction, 77.6 per cent.; current density, 6.54 amp. per square foot; voltage, 1.23; production of copper, 1.72 lb. per kilowatt-hour; H_2SO_4 in solution, 2.73; sulphur per pound of copper, 1.06 lb.; total days run, 40.2; total cathodes produced, 803 lb.

PROCESS OF POPE AND HAHN

While these tests were being carried on, another process was being developed at Raritan by F. A. Pope and A. W. Hahn. The Antisell idea was in a sense a combination of diaphragm and depolarizer, without any attempt to remove the iron. Pope and Hahn planned to remove the iron (and most of the other impurities as well) before sending solution to be electrolyzed.

Their process was cyclic. Ore crushed to 4-mesh was leached for 3 days with dilute sulphuric acid, the counter-current leach being so distributed as to permit of drawing off two classes of solution—one high in iron, the other low. The high iron solution was heated to about 200° F. with the addition of the requisite amount of finely ground (90 per cent. through 200-mesh) copper oxide in the form of roasted ore. The mixture of solution and oxides was agitated for 3 or 4 hr. and then put through a filter press. The result was the removal of nearly 90 per cent. of the total iron and 70 to 80 per cent. of the alumina from solution. An electrolyte high in copper and low in all impurities was produced and the precipitate, as filter cake, was granular and easy to wash and handle.

The other portion of the leaching solution, low in iron, was electrolyzed and returned to the cycle until its iron content was high enough to require purification.

Results of the Pope-Hahn process were very satisfactory, but the plant had hardly started when the war broke out and all experimental work was stopped. The following summary will indicate the results obtained: Ore leached, 329 tons, 10 charges; average copper heads, 1.12 per cent.; copper tails, 0.30 per cent.; water, tails, 9 per cent.; copper per kilowatt-hour, 1.0 lb. About 40 lb. of roasted Miami concentrate per ton of ore handled took care of impurities; 92 per cent. of the copper in the concentrate was recovered.

Working the Pope-Hahn process would have required the installation of roasting plant, fine-crushing plant, agitators and filter presses and the purchase of high-grade sulphide ore or concentrates. It offered decided advantages, among them a practically pure solution for electrolysis, together with the regeneration of a high percentage of the acid necessary for leaching.

GREENWAY PROCESS

Before the study of the effect of impurities had progressed very far, an interesting fact was noticed. If neutral solution from the leaching system was circulated through fresh ore, a large part of the iron in solution was precipitated, presumably as a basic sulphate. This basic sulphate was apparently not readily soluble in a solution containing more acid. A possible method of purification from iron seemed to be indicated and the results of small-scale tests were so encouraging that a 1-ton plant was built. In the first tests, ore crushed to about ½ in. was leached for 6 days. The oldest charge of ore was in contact with fresh leach from the electrolytic cells. This leach carried about 3 per cent. of free H_2SO_4. Solution passed in counter-current through six tanks containing ore, and the acid of the leach was neutralized by the time it reached the fourth or fifth tank. In the last two tanks a neutral solution was in contact with fresh ore and here the iron was deposited.

The results obtained will be evident from the data of Table 2. The ferric iron was under good control for more than 100 complete cycles.

TABLE 1.—*Analysis of Average Cornelia Oxidized Ore*

	Per Cent.
SiO_2	65.7
Al_2O_3	15.3
Fe total	4.5
CaO	0.6
MgO	1.6
MnO	0.14
Cu	1.50
	Oz. per Ton
Au	0.005
Ag	0.20
	Per Cent.
NaO	3.6
K_2O	4.6
P_2O_5	0.3
Cl	trace
CO_2	1.3
	99.04

At this time it was already practically settled:

1. That a good extraction could be obtained with dilute sulphuric acid.

2. That the acid consumption was not prohibitive.

3. That electrolytic copper could be produced successfully.

There remained, however, some important problems to be solved. Among these were:

1. Crushing; from two points of view: (1) for percolation in a deep bed; (2) for extraction.

2. Further study of the fouling of the solution, including the general accumulation of soluble salts, but especially those which might affect extraction or power efficiency in electrolysis.

3. Circulation and general handling on a small commercial scale.

4. Any factors other than those under (2), which might affect electrolysis itself or the quality of copper produced.

TABLE 2.—Summary of

Charge No. (Inclusive) Date	1–19 8–8 to 8–31	20–74 9–1 to 10–19	75–100 10–31 to 11–25	101–116 11–26 to 12–10	117–217 12–11 to 3–27	218–259 4–20 to 5–31	260–275 6–1 to 6–16	276–299 6–17 to 7–11	300–330 7–12 to 8–11	341–413 8–18 to 11–3
Average:										
Heads—per cent copper	1.70	1.22	1.55	1.39	1.29	1.30	1.36	1.29	1.22	1.33
Tails—per cent copper	0.64	0.29	0.43	0.32	0.24	0.24	0.19	0.17	0.24	0.27
Extraction—per cent	62.8	75.9	72.2	77.1	81.4	81.5	85.8	86.8	80.3	79.7
Per cent. ore on 4-mesh	29.0	47.0	43.4	28.0	33.0	25.0	26.0	25.1	35.1	39.0
Days leached	6.0	6.0	6.0	6.8	7.6	8.0	7.5	7.5	7.5	8.0
Circulation—1-1 gal. per ton per min	2.0	1.8	3.3	3.3	3.0	2.0	2.0	2.0	2.0	2.0
Advance—1-1 gal. per ton per min	0.42	0.40	0.17	0.20	0.20	0.16	0.16	0.20	0.16	0.29
Electrolytic data:										
Current density, amp. per sq. ft	9.8	12.0	7.0	7.3	7.1	6.9	9.2	6.6	5.3	8.5
Volts—drop in tank	2.11	2.36	2.00	2.15	2.12	2.29	2.30	2.08	2.03	0.97
Lb. copper per kw.-hr	0.98	0.83	1.08	1.01	0.99	0.77	0.76	1.07	0.95	2.0
Ave. free H_2SO_4 in cells	4.2	3.0	2.8	2.7	2.9	2.9	3.3	3.4	2.8	3.8
Ave. circulation in cells	3.8	4.6	4.4	4.2	5.8	7.0	8.2	7.5	7.8	5.0
Ampere efficiency	83.7	76.8	83.5	85.0	83.1	67.2	67.8	75.0	74.0	69.5
Acid consumption:										
Total lb. 100 per cent. H_2SO_4 used	1,217.0	1,160.0	729.0	467.0	3,428.0	1,107.0	492.0	1,002.0	1,113.0	4,240.0
Acid per lb. cathode Cu.	5.33	1.76	1.16	1.62	1.66	1.46	1.30	2.01	2.74	6.65
Acid per lb. Cu leached	3.02	0.94	1.06	1.36	1.61	1.24	1.31	1.86	1.83	2.30

NOTES.—Charges Nos. 1 to 19 had no crushing plant, so screened rejects from sampling oxide pits were used. Ore had excess fines and contained some sulphides. No filter bottoms in tanks. Segregation, started with new solution. Leaching started Aug. 8, electrolytic end started Aug. 17 with grid lead anodes.

Charges Nos. 20 to 74, on Oct. 6, tried reduction with liquid SO_2 using filtros bottom to distribute gas. Test stopped Oct. 14.

Charge No. 64.—All other charges normal. On Oct. 19, decided to discard one-half of solution volume, dilute to normal volume with well No. 1 water and make special run

Charges Nos. 75 to 100.—No SO_2 used and water from well No. 1 (high Cl) used on these charges.

Charges Nos. 101 to 116.—Same solution, longer leach, finer crushing variable conditions—No SO_2.

Charges Nos. 117 to 217.—On Dec. 14, SO_2 reduction tests were begun under the direction of F. L. Antisell. Lead-coke anodes were used. Tests discontinued Dec. 26—between charges 117 to 127. Leaching varied from 7 to 13 days. Charges Nos. 127 to 217 were run under normal conditions.

Charges Nos. 162, 163, 164 were high-grade ore—"k-10"-av. 2.7 per cent. Cu. These have not been included in the above averages. On Feb. 27, the average current density of 7 amp. per square foot was reduced to 4 amp. per square foot.

Charge No. 218.—April 19 was started with 40 tons high ferric iron solution, other conditions normal.

Charge No. 275.—June 17, arrangements made for another SO_2 run with lead anodes; same solution used. Test continued until charge 300.

Charges 300 to 330 run without SO_2 and under normal conditions.

Charge Nos. 341 to 413.—New solution, carbon anodes, SO_2 reduction, air agitation.

Ore used on charges Nos. 20 to 413 was from open cut and averaged 1.307 per cent. Cu.

The 1-ton plant was therefore continued in operation and a plant to handle 40 tons of ore per day was built.

It is fortunate that this larger unit was built, for a good many things that had seemed easy to do in the 1-ton plant refused to work at all in the 40-ton. Fouling by iron and aluminum salts kept on increasing, the ferric-iron content of the solution rose to a point where the ampere efficiency in electrolysis was poor, and various undesirable phenomena appeared.

So the 40-ton plant was kept in continuous operation for nearly a year and all the factors that had been studied in the 1-ton plant were taken up, one at a time, and worked over again until finally it seemed that they could all be brought under complete control.

Final Leaching Process

The final process decided upon was as follows: Ore crushed to about 4-mesh; leached 8 full days by counter-current; washed with three counter-current wash waters (or possibly four); solution, practically neutral during last 2 days of contact with ore, is sent through reduction towers, where it meets sulphur dioxide in counter-current. The ferric iron is thus reduced to below 0.4 per cent. Thence it goes through a revolving tumbler in contact with cement copper. There the ferric iron is still further reduced and a corresponding amount of cement copper passes into solution. From the cement-copper tumbler the solution passes to a settling pond and then into the electrolytic cells, where a part of the copper is removed; then back into the leaching system, passing first through the oldest ore which has already been leaching for 7 days, and so on to begin the cycle again.

Important Factors

In a cyclic process such as this, no single factor or step can be said to be the most important. We can, however, consider a list of those factors that were studied with especial attention and can offer definite figures on the results to be expected.

Extraction

Both in the 1-ton and the 40-ton plants, it was easy to obtain 80 per cent. extraction, using about 3 per cent. sulphuric acid and 8 days of counter-current leaching by upward percolation on 4-mesh material. An extraction of 83 per cent. was reached over considerable periods of time. The continuous attainment of this figure was limited by other factors than the nature of the ore or the size to which it was crushed, such as the fouling of solution, with the result that salts carrying copper actually crystallized in the ore in the leaching tanks and were not completely dissolved by washing.

Incomplete washing was, of course, another reason for low extraction under otherwise normal conditions, and the possibility of improvement in this point is clearly shown by the wash-water data of Table 6.

Keeping well within the time limit for washing set by the fixed cycle, it should apparently be possible to get an extraction of 82 per cent. or better.

Acid Consumption

The 1-ton plant, over its whole life of 413 cycles and under all sorts of favorable and unfavorable conditions, showed a net consumption of 1.65 lb. of 100 per cent. H_2SO_4 per pound of copper leached. This was somewhat lower than the figures given by the preliminary tests on the ore.

The 40-ton plant apparently did not do so well. During its 301 cycles, the average net consumption was 2.8 lb. of acid per pound of copper leached. The difference in favor of the 1-ton plant is probably due to the large leakage and general losses in the 40-ton plant.

In designing the plant which is now being built at Douglas to furnish acid for Cornelia, 3 lb. has been taken as a conservative figure for acid consumption.

Power Consumption for Electrolysis

In the 1-ton plant, with lead anodes, the average was 0.934 lb. copper per kilowatt-hour at 8 amp. per square foot of cathode surface. This was for all the conditions, good and bad, under which the plant was operated. With graphite anodes, also under all sorts of conditions, the figure was 2 lb. copper per kilowatt-hour at 8.5 amp. per square foot.

In the 40-ton plant, with lead anodes, at an average current density of 6.9 amp. per square foot, the average production under all conditions was 0.87 lb. copper per kilowatt-hour. With graphite anodes, the figure was 1.65 lb. It should be said, however, that the ferric iron was never under proper control during these tests with graphite in the 40-ton plant. During series VI of the 40-ton plant, with current density 6.4 amp. per square foot, the copper yield was 1.04 lb. per kilowatt-hour. This run was with lead anodes and under fair conditions, so far as ferric iron was concerned. In the final run with lead anodes, and with both ferric iron and general fouling under still better control, an even higher figure was obtained.

Lead anodes will be used in the big plant. The reasons for this choice will be given later.

TABLE 3.—*Summary of All 40-Ton Tests with Lead Anodes. Feb. 1 to Nov. 6, 1915*

Charge No. (Inclusive) Date	1 to 81 Feb. 1–May 21	91 to 130 June 1–July 12	131 to 154 July 13–Aug. 4	155 to 170 Aug. 5–Aug. 22	171 to 201 Aug. 23–Sept. 25	202 to 238 Sept. 26–Nov. 7	1 to 238 Feb. 1–Nov. 7
Average:							
Heads—per cent copper	1.248	1.314	1.225	1.274	1.286	1.361	1.281
Tails—per cent copper	0.294	0.226	0.242	0.265	0.346	0.223	0.275
Extraction—per cent	76.4	82.8	80.2	79.2	73.1	83.6	78.5
Per cent. ore on 4-mesh	25.0	25.3	35.0	32.0	40.7	37.9	36.0
Days leaching	9.3	8.0	8.0	8.0	8.0	8.0	8.3
Circulation—gal. per ton per min	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Advance—gal. per ton per min	0.20	0.35	0.35	0.35	0.35	0.38	0.30
Electrolytic data:							
Current density—amp. per sq. ft	7.4	7.1	7.1	7.1	6.0	6.4	6.9
Volts—drop in tank	2.43	2.20	2.18	2.04	2.18	2.07	2.24
Lb. copper per kw.-hr	0.72	0.99	0.85	0.98	0.86	1.04	0.87
Ampere efficiency	66.8	83.3	70.8	76.5	71.7	82.3	74.0
Lb. cathode copper	49,775.0	30,652.0	17,431.0	10,521.0	18,273.0	23,717.0	150,349.0
Acid consumption:							
Total lb. used—100 per cent	188,484.0	98,442.0	47,484.0	37,197.0	85,173.0	102,402.0	559,182.0
Lb. H_2SO_4 per lb. cathode Cu	3.82	3.21	2.72	3.53	4.66	4.31	3.71
Lb. H_2SO_4 per lb. Cu. in solution	2.88	2.69	2.39	2.58	3.67	2.98	2.89
Sulphur:							
Total lb. used	None	None	4,693.0	5,434.0	18,767.0	26,697.0	55,861.0
Lb. S per lb. cathode Cu	None	None	0.27	0.52	1.02	1.12	0.80
Lb. S per lb. iron reduced	None	None	1.50	0.56	1.20	0.99	1.08
	Old solution– No SO_2 first run	New solution— No SO_2 new run	SO_2 small tower acid electrolyte	SO_2 small tower neut. electrolyte	SO_2 large tower solution saturated	SO_2 large tower solution diluted	Av. of all 40-ton tests with Pb anodes

NOTE.—Forty-ton plant started Feb. 1, 1915, with Pope-Hahn solution, tank house started Feb. 26. Control of ferric iron lost on Mar. 12

Charge No. 26, various attempts made to change iron ratio. Low tank-house efficiency caused copper content of leaching solution to increase to such an extent as to interfere with the extractions. Troubles with pumps, talcy ore, linings, etc. Solution of these first 81 charges discarded on May 21 and a new series of tests started, May 22, 1915. Charge No. 91, new solution was made up for this run. Electrolytic plant started June 1. Iron control lost on charge No. 117. June 27, reduction of the ferric iron decided upon and on July 13 the acid electrolyte was circulated through small reducing tower. As reduction did not appear to gain on the oxidation in the tank house it was decided to circulate neutral solution through the tower. Results showed reduction of ferric iron in neutral solution to be less difficult than the ferric iron in acid solution. Large tower built, and completed Aug. 23. Acid electrolyte reduction continued through large tower. Solution becomes saturated with salts and on sudden changes in temperature deposits crystals on ore. Extraction decreases and it was found necessary to dilute the solution by intermittently "bleeding" the "high iron" tank and making up the deficiency with wash water. The average extraction on the 15 charges before bleeding was 68.7 per cent; the average extraction on the first 15 charges after bleeding, 80.7 per cent. On Nov. 7 carbon anodes were put into the tanks.

Control of Ferric Iron

Quite early in the study of the conditions for good electrolysis it became evident that the ferric-iron content of solutions must be kept within a low limit. The 1-ton plant took care of itself over a long period of time, as far as the ferric-iron content and the general fouling were concerned, without the use of sulphur dioxide or any other outside reagent. Iron and alumina were apparently precipitated on the ore.

The 40-ton plant started off in the same way. But before long the ferric iron began to go up and the ampere efficiency to go down. This is evident from the table showing general results on the 40-ton plant. Series II began with fresh acid and lasted 40 days. During this period 0.99 lb. Cu per kilowatt-hour was the electrolytic yield. The current efficiency during this period of low ferric iron and clear solution was 83.3 per cent. During period III the same solution was continued in use, but the ferric iron was not under control, and kept increasing. The analysis of the solution on July 22, in the middle of this run, is shown in column III of Table 4. The ferric iron content had risen rapidly from 0.180 per cent. on June 20 to 1.222 per cent. on July 22. The result was that during period III, the current efficiency was only 70.8 per cent. and the yield was 0.85 lb. copper per kilowatt-hour. From this time until the

TABLE 4.—*Solution Analyses*

Second Period of 40-Ton Plant Operation Beginning with Charge 91 in 1915

	I 6–1, Per Cent.	II 6–20, Per Cent.	III 7–22, Per Cent.	IV 8–22, Per Cent.	V 9–23, Per Cent.	VI 11–5, Per Cent.
Cu	1.11	2.26	2.50	3.39	3.47	2.95
H_2SO_4 free	4.03	2.88	2.56	3.15	2.40	3.05
Fe^{++}	0.09	0.176	0.720	1.49	2.14	2.52
$FeSO^{+++}$	0.164	0.180	1.220	1.07	0.80	0.239
Al_2O_3	0.145	1.250	2.730	3.69	4.40	2.840
CaO	0.092	0.090	0.080	0.050	0.120	0.018
MgO	0.132	0.122	0.120	0.150	0.049	0.054
MnO	0.023	0.030	0.030	0.050	0.046	0.029
Cl	0.08	0.08	.....	.....	0.130	0.090
SiO_2	0.075	0.084	0.110	0.040	0.960	0.096
Specific gravity	1.100	1.150	1.250	1.370	1.376	1.330

Column I gives solution analysis at beginning of a run, where fresh acid has been used, and before electrolysis has been started.

Column III is the analysis of solution when reduction of ferric iron was begun.

Column V shows solution at maximum concentration. Cold nights caused the separation of a large amount of $FeSO_4$, $CuSO_4$, $24H_2O$.

Column VI is an analysis of solution under normal conditions, after "bleeding" regularly for about 40 days.

end of the experimental work, sulphur dioxide gas was used regularly to control the ferric iron. With lead anodes a low content of ferric iron is not necessary. The solution may contain 0.3 or 0.4 per cent. without any marked effect on the current efficiency.

Sulphur Consumption

To maintain satisfactory control of the ferric iron was, at this point in the work, the greatest apparent difficulty. To follow the course of the tests it will be necessary to keep in mind both Tables 3 and 4. The first of these shows the conditions under which the 40-ton plant was operating; the latter, the analysis of the solution at various times.

During period III, the acid electrolyte was circulated through a tower about 2 ft. square, in contact with sulphur dioxide. The acid electrolyte was taken from the electrolytic system and returned to the same system. The weight of sulphur that could be burnt and brought in contact with solution as SO_2 was too small, and the ferric iron kept on increasing. Under these conditions, we were burning 1.5 lb. sulphur per pound of ferric iron reduced.

Series IV began with only a single change, but this proved to be a very important one. The *neutral* advance from the leaching system was circulated through the SO_2 tower before entering the electrolytic system. Under these conditions, only 0.56 lb. of sulphur was required per pound of ferric iron reduced.

Although the small tower was not of sufficient capacity to make control easy, by Aug. 22 the ferric iron was on the down grade—1.07 per cent. as against 1.22 on July 22.

From this time on, as long as lead anodes were in use, the ferric iron did not get out of control, and on Nov. 5 the solution showed only 0.24 per cent. of this troublesome substance. The results with graphite anodes will be considered later.

Fouling of Solution

In the 40-ton plant the entire solution began to foul badly after about 100 charges had been leached. The cooler weather was beginning at this time and a bulky double sulphate of iron and copper crystallized out through the ore in the leaching tanks and plugged all the pipes and launders. To remedy this, a portion of nearly neutral solution was drawn off continuously from the leaching system and passed over scrap iron, making cement-copper. It was found that treating 1 per cent. of the total bulk of solution each day would maintain the fouling at a constant low point where it could cause no trouble. Columns V and VI of Table 4 show the effect of this "bleeding" of solution.

The cement-copper so produced was returned to a wooden tumbler and electrolyte on the way to the cells passed through it. Ferric iron is reduced to ferrous in this way and copper passes back into solution.

Under these conditions only electrolytic copper is produced and at the same time the control of the ferric iron is aided considerably. In the 40-ton plant it was apparently necessary to send about 12 per cent. of the total copper through this side cycle and back into the main one, to control completely the fouling materials introduced from the ore during leaching.

Grade of Copper Produced

Cornelia copper produced direct from the ore is the equal of any electrolytic copper produced in this country. Fortunately the orebody contains only minute amounts of either arsenic, antimony or bismuth. The analysis and the physical tests on Cornelia cathodes will be found in Table 5.

Table 5.—Analysis of Cornelia Copper Cathode

	Top, Per Cent.	Bottom, Per Cent.		Top, Per Cent.	Bottom, Per Cent.
Cu	99.900	99.868	S	0.0369	0.0655
Fe	0.0186	0.0291	As	0.0013	0.0017
Cl	0.0012	0.0019	Sb	0.00095	0.0014

Physical Tests on Wire Bars

Tensile test	64,500
Elongation	1.9
Bends	36
Twists	43
Conductivity	100.7

Size of Material. Percolation

The work at Ajo was all done on the product from a Symons fine crusher, without any screening (except for tests on sized material). It was nominally a 4-mesh product but carried an average of about 30 per cent. of oversize. Extraction on this is satisfactory and while it could be increased by finer crushing, a large percentage of the copper minerals lies in fracture planes, and no very great profit would result.

This product is especially well suited to upward percolation in a bed 12 ft. deep or more. It does not segregate nor channel badly, drains well and is easy to wash.

Circulation in the Leaching System

In the 40-ton plant, each of the eight active tanks had a closed circulation, upward through the ore, of about 80 gal. per minute. This rate is probably much higher than necessary, but for the experimental work it was made ample and retained as a fixed factor. Beside this closed circuit, which runs continuously, a portion of the solution from each tank is continuously advanced to the next tank in the series.

Fresh electrolyte from the cells, low in copper and of maximum acid content, leaves the electrolytic system and passes into the leaching tank containing the oldest ore—that which has been leaching for 7 days. Practically neutral solution, high in copper, leaves the leaching system at the tank containing new ore. Between these two points, and rigidly linked to them, is the advance from tank to tank.

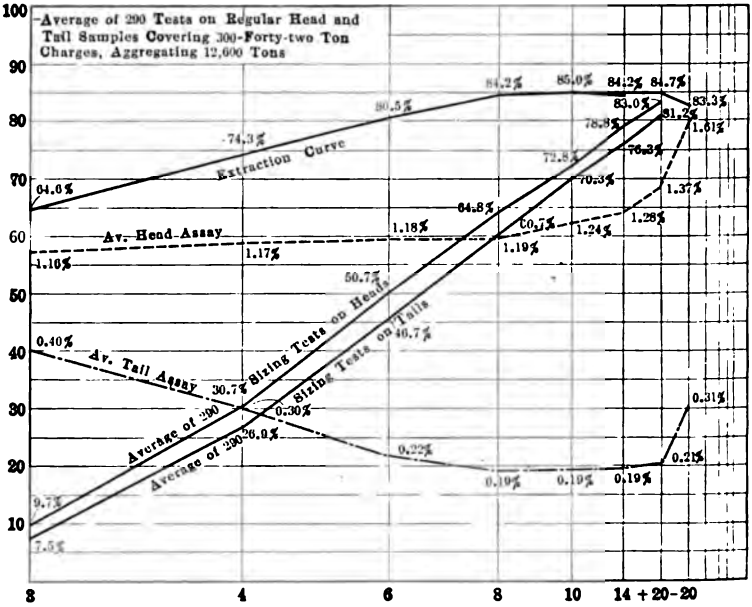

FIG. 1.—SIZING TESTS ON NEW CORNELIA OXIDIZED ORE SHOWING ALSO AVERAGE ASSAYS ON HEADS AND TAILS AND AVERAGE EXTRACTIONS ON SIZED MATERIAL.

It is of great importance to have a practically neutral solution leaving the last leaching tank and starting back toward the electrolytic system.

At the same time, we want as high an extraction as possible, which means as many tanks as possible with a fairly high acid content. To attain both these objects at the same time the advance from tank to tank is carefully regulated so that about six tanks of the eight active ones are acid practically all the time. The two others are neutral and the point of neutrality of the system just balances, varying possibly one tank in either direction during 24 hr., but returning to its original position.

Circulation in the Electrolytic System

The electrolytic cells, together with tanks and sumps for storage, form a second closed circuit from which solution is bypassed continuously into the leaching system at the same rate as the advance from cell to cell. To produce good, coherent cathodes, the circulation through the cells must be much greater in volume than the advance from the leaching system, as the latter is rigidly fixed by the necessity for maintaining neutral leaching tanks. While the advance in the 40-ton plant was about 12 gal. per minute, it was advantageous to circulate in the tank house at rates up to 150 gal. per minute.

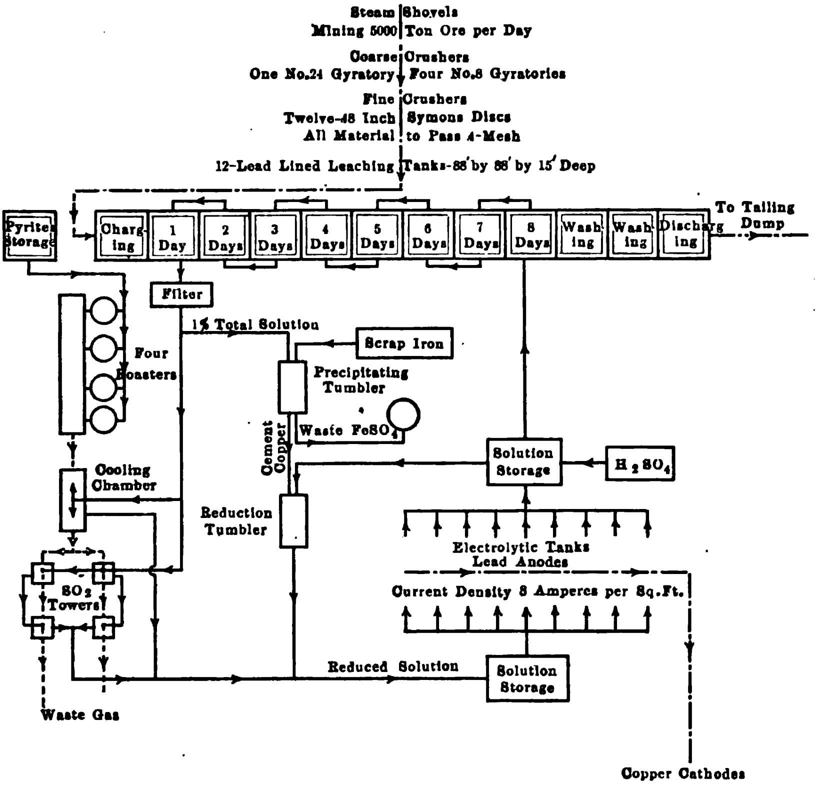

FIG. 2.—FLOW SHEET OF 5,000-TON PLANT OF NEW CORNELIA COPPER CO.

This circulation gives satisfactory results with lead anodes, but when graphite is used, rapid circulation alone is not enough to give good depolarization at the anode. Violent agitation in the electrolytic tanks is necessary to attain the lowest voltage when graphite anodes are in use, and this was accomplished by blowing air through small holes in lead pipes laid along the bottom of the cell.

Lead vs. Graphite Anodes

In making final decision on the process to be used, there were many mechanical questions to be settled. Crushing, method of percolation, circulation, pumps, structural materials, the handling of ore and tailings and many other points came up for decision. Besides these, there were some questions which bear more directly on fundamental principles. One of these was whether we should use lead or graphite as material for anodes.

Lead was in service for over a year and its behavior was well understood. But graphite promised much better results as far as power efficiency is concerned, and also as a means of regenerating a much larger part of the necessary acid for leaching.

The 1-ton plant gave most encouraging results with graphite over a period of continuous operation of nearly 3 months. With a current density of 8.5 amp. per square foot, over 2 lb. of copper per kilowatt-hour was produced, and during part of this time conditions were not good. The ferric iron was not under complete control. When all conditions were right, we made 2.25 to 2.40 lb. copper per kilowatt-hour; on some days as high as 2.75 lb. This was during the warm weather at Ajo, when electrolyte temperatures held well above 105° F. The 40-ton plant was using the large tower for reduction and the 1-ton plant used the small one. The SO_2 used in controlling ferric iron was bypassed from the main supply.

The 1-ton electrolytic plant was out of doors and while the odor of SO_2 was strong in the immediate neighborhood of the cell, it was not bad enough to prevent working over it.

By the time the graphite anodes were installed in the 40-ton plant, the average electrolyte temperature was only about 75° F. We could now determine accurately the efficiency of absorption and reduction by SO_2 (which we could not do in the 1-ton plant) and we could also examine carefully the circulation conditions necessary to control ferric iron.

The problem now began to look harder. As measured by the production of ferric iron from ferrous in the cell, the anodic efficiency of graphite, with rapid air agitation, is well over 100 per cent. With lead it is 30 to 35 per cent. We had therefore to combat about three times as much ferric iron per pound of copper with graphite as with lead.

Under the existing temperature conditions and with the rapid circulation that was necessary in the electrolytic circuit, the reduction efficiency of the SO_2 was only about 20 per cent. We could not control the ferric iron without either allowing the electrolyte to stand for a good many hours between the SO_2 towers and the cells, or else heating it to about 150° F. during its passage from towers to cells.

The absorption of SO_2 in the towers was perfectly satisfactory. But unless it was used to better advantage in the reduction of the iron in the solution, it was impossible to live in the tank house without either very careful ventilation or a diver's outfit. With the violent air agitation that was necessary for high power efficiency, most of the SO_2 which had been absorbed in the towers was blown out in the tank house.

These difficulties are by no means insuperable. Of more fundamental nature is the low current efficiency obtained with graphite and its extreme sensitiveness toward ferric iron. When ferric iron is practically completely removed by careful reduction with SO_2, graphite can be made to give high current efficiencies. But the very slightest rise in ferric-iron concentration results in an immediate and considerable lowering in ampere efficiency. Even in our good runs with graphite in the 1-ton plant, the current efficiency was not much above 70 per cent. With lead anodes, and without any particular attention to the ferric iron, which was usually 0.3 or 0.4 per cent., we often obtained current efficiencies of 90 per cent. or better over considerable periods.

It may be of interest to state some of the points that enter into a decision between the two kinds of anodes:

1. The first cost of an installation is about the same.
2. The life of lead is known with some accuracy. The life of graphite in this service is not known.
3. The salvage value of lead is high. With graphite it is practically nothing.
4. The probable power saving to be expected (graphite over lead) if graphite were successfully operated, is about 0.6 kw.-hr. per pound of copper.
5. More than twice as much acid is regenerated with graphite as with lead.
6. Circulation must be more rapid with graphite.
7. Agitation during electrolysis is absolutely necessary with graphite.
8. The solution must stand for some time (probably 16 to 24 hr.) or be heated to 150° F. or higher, to obtain even reasonable SO_2 efficiency with graphite. This is not necessary with lead (see note under 10).
9. Much larger towers are needed when graphite is used.
10. The volume to be pumped to the top of the towers is larger with graphite.

(These last two statements contain the fact that acid electrolyte must be sent through the towers. The iron content of the solution in the cells must not rise above 0.2 per cent., if good efficiency is to be reached with graphite. With lead anodes it may go to 0.4 or 0.5 per cent. without much of any effect on power consumption. With lead it is sufficient to send the neutral advance to the towers. With graphite not only this volume, but also a much larger volume from the tank-house circulation which must be kept low in ferric iron must be sent.)

11. The tank house for graphite must be carefully ventilated.

12. The tank house for graphite must be larger (about as 9 to 7) on account of the lower current efficiency.

13. Operation with graphite requires the closest attention to every detail of reduction, circulation and agitation. Operation with lead is practically fool-proof.

This last point is probably the most important of all. There are enough so-called "minor" troubles about a leaching plant.

Control of Ferric Iron in Acid and Neutral Solution

During the periods when acid electrolyte was being sent to the tower for the reduction of ferric iron by sulphur dioxide, the sulphur efficiency was low. It proved to be about 20 to 25 per cent. Two causes at least combine to produce this result: First, the low solubility of SO_2 in acid solution, and second, the slowness of the reduction reaction in acid solution. Neutral solution was sent to the towers in later runs and on this the sulphur efficiency reached 75 to 80 per cent. That is to say, of the actual weight of sulphur burned to SO_2, 75 per cent. is utilized in the reduction of ferric iron to ferrous.

Washing the Ore

Each charge was given three washes. The first was the second wash of the preceding charge; the second, the third wash of the preceding charge; the third, fresh water. The wash which had been used three times was run into the electrolytic system. One complete fresh wash water was just sufficient to balance the loss from the system in the tailings and by evaporation.

The summarized wash-water data are given in Table 6 for the solution which was started in the 40-ton plant May 22, 1915. The most significant thing in this table is the effect of fouling as shown in the second column. While the solution was fresh, washing in the prescribed manner, the extraction was 81.1 per cent. and soluble copper left in the tailings was 0.045 per cent. When the solution became foul, even though a little more fresh water was used, extraction dropped to 73.5 per cent. and the soluble copper in the tailings was 0.103 per cent. Later, when fouling was being reduced by "bleeding" and additional wash water could be used in the same cycle, the extraction rose to 83 per cent. and the soluble copper in the tails dropped to 0.033 per cent.

It is of course possible to wash tailings with extra water, passing this over scrap iron and making cement-copper, and this should naturally be done until the expense of operation balances the value of the recovered copper. The experiments at Cornelia were carried no further than is shown in this table.

All the experimental work in the 40-ton plant was carried out during continuous operation, so that the result should correspond as closely as possible with commercial conditions. The regular tonnage was crushed each day and charged into the leaching tank. Tailings were removed, and circulation, advance, washing, electrolysis and reduction carried on without a break. Any change in operation was carried on for at least a week, and usually for 10 days. If the change promised to give improved results, it became a fixed part of the regular operating routine and the next point came up for study.

TABLE 6.—*Wash-Water Data*

Charges Covered by Period	No. 91 to No. 170, May 22 to Aug. 20	No. 171 to No. 205, Aug. 21 to Sept. 25	No. 206 to No. 301, Sept. 26 to Jan. 10
Tons of ore treated	3,376	1,435	4,051
Per cent. copper in the heads	1.278	1.282	1.331
Per cent. copper in the tailings	0.240	0.342	0.237
Per cent. extraction	81.1	73.5	83.0
Per cent. copper in washed tailings	0.195	0.239	0.204
Per cent. extraction if tailings had been washed clean	84.7	81.3	85.3
Average moisture in tailings	10.52	9.57	9.65
Average number of washes	3	3	3
Total gallons new water used per ton of ore leached	40.6	42.2	59.1
Gallons entrained in tailings per ton tailings discharged	24.2	21.2	21.8
Gallons unaccounted for per ton of ore leached, *i.e.*, evaporation and leakage, etc.	16.4	21.0	12.1
Gallons of solution bled per ton ore	None	None	25.2
Specific gravity of 8th day solution	1.100 – 1.290	1.290 – 1.400	1.400 – 1.300
Specific gravity of last wash	1.050 – 1.060	1.060 1.150	1.150 – 1.050
Average per cent. copper in first wash	1.66	2.96	2.25
Average per cent. copper in second wash	1.17	2.17	1.45
Average per cent. copper in third wash	0.61	1.25	0.67
Average per cent. H_2SO_4 in third wash	0.35	0.35	0.35
Pounds soluble copper per ton tailings	0.9	2.1	0.7
Probable extraction if one more wash had been used and half of this soluble copper had been recovered. Per cent.	83.3	79.0	84.2

Average of whole run:

Per cent. copper in heads	1.338
Per cent. copper in tailings	0.256
Per cent. extraction	80.7
Per cent. extraction—according to washed tails	84.1
Per cent. extraction—if 4th wash had been used	83.0
Per cent. moisture in tailings	10.0
Average gallons of solution entrained per ton of tails	22.6

Using this method, when we were through, we had a definite set of conditions under which known results could be produced for any length of time. There were no odds and ends to go back and pick up. No "minor" problems, which might turn out on investigation to be fundamental, remained to be solved. Mr. Greenway saw that this plan was strictly carried out.

During the experimental work, something over 180,000 lb. of perfect cathodes were produced, besides a good deal of cement-copper.

SUMMARY

So far as the Ajo tests can show what will happen in a 5,000-ton plant, the following may be expected:

Extraction on 1.4 per cent. ore using closed wash cycle, 82 per cent.

Extraction on 1.4 per cent. ore using extra wash water, 83 per cent.

(The extraction figure used in making estimates has always been 80 per cent.).

Acid consumption (100 per cent. H_2SO_4), 3 lb. per pound of copper. "Bleeding" to prevent fouling of solution.

Power efficiency in electrolysis with lead anodes, 1 kw.-hr. per pound of cathode copper.

Sulphur consumption, 0.5 lb. per pound of cathode copper.

Neutral electrolyte only will need to pass through the reduction towers.

About 1 per cent. per day of the total solution volume must be sent over iron to control fouling.

About 12 per cent. of the total copper produced will pass through the cement-copper side cycle.

The circulation used in the test-plant leaching tanks (2 gal. per minute per ton of ore) is undoubtedly much higher than necessary. It is probable that a small part of this volume would give equally good extraction results.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at th Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Advent of Flotation in the Clifton-Morenci District, Arizona

BY DAVID COLE,* EL PASO, TEXAS

(Arizona Meeting, September, 1916)

At the time flotation appeared upon the metallurgical horizon in Arizona, the writer, under the direction of Dr. Ricketts, was engaged in remodeling and enlarging the No. 6 Concentration Plant of the Arizona Copper Co. at Morenci, and the work had been in progress nearly a year before the Inspiration experiments with flotation had disclosed the revolution in concentration that was at that time impending.

The simplified flowsheet worked out for the remodeling of the Morenci plant had been based upon the removal of the freed metal in a minimum number of stages by treatment upon tables equipped with Butchart riffles, the latter being adapted to accomplish both classification of feed and removal of the metal at one operation, substantially as described in my previous paper.[1]

The scheme of treatment for the slime was based upon the well-known fact that after copper sulphides, such as chalcocite and chalcopyrite, etc., are reduced to a certain extremely fine state of comminution they are definitely beyond the reach of separation upon any of the concentrating devices then known.

By using drag-belt classifiers (which served as conveyors as well as separators), the overflow would be of the usual "slimes" class. Experiment had shown that when these drag-belt overflows were properly diluted (that is, to about 5 to 7 per cent. solids in the feed under treatment) the very fine sand, and especially the very fine but still granular sulphide particles, would, if given a short distance to fall, quickly settle out in prime condition to yield an excellent recovery on vanners; a further valuable feature would be that the very fine non-separable final overflow tonnage might be discharged direct to tailing, thus conserving space requirements so as to permit within the old building, without embarrassment, the increase of capacity required.

For reasons that are obvious, this kind of feed preparation for the slime could not be successfully accomplished in any form of pointed boxes or spitzkasten device, therefore a further elaboration of the drag-belt idea was worked out as the best method for accomplishing the separation. This machine was known as a colloid separator and is shown in Figs. 1 and 2. It works on the premise that nearly all of what may be called ponderable material in the thinned pulp, falling but 2 in., will lodge upon the belts and be quickly removed, while the flocculent slime material will remain in suspension and go away with the overflow. In this way a feed is prepared for the vanners containing a maximum amount of the very fine but granular sulphides and a minimum of colloidal material; at the

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* Consulting Engineer.

[1] Development of the Butchart Riffle System at Morenci, *Trans.*, vol. 51, p. 405 (1915).

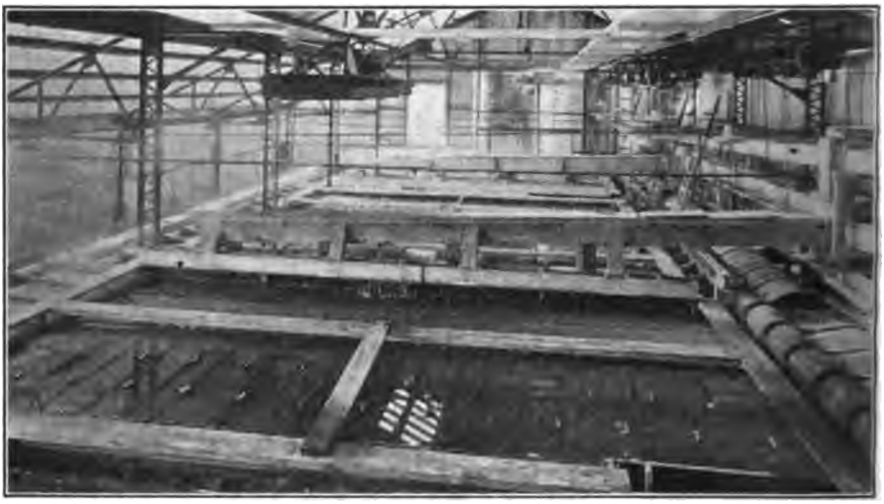

Fig. 1.—Drag-Belt Separators in Mill of Arizona Copper Co.

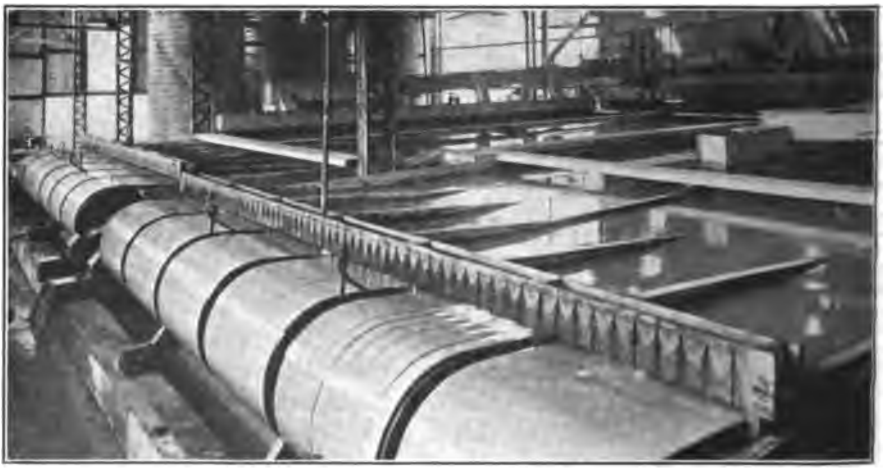

Fig. 2.—Another View of Drag-Belt Separators.

same time the drag-belt overflow product contains a minimum of sulphide particles and a maximum of flocculent slime or colloidal pulp. This final overflow was found in practice to be approximately two-thirds of the total tonnage handled by the belts and treatment upon vanners was, as far as copper recovery was concerned, devoid of beneficial results. The copper escaping in this overflow was in the form of extremely fine chalcocite, bornite with a little pyrite and chalcopyrite, together with oxidized, and water-soluble, copper salts. Taken together, these gave the overflow a copper tenor of from 1 to 1.3 per cent. in a ton of dry material, thereby accounting for the larger part of the tailing losses. However, this overflow, with its ultrafine and otherwise handicapped copper-bearing material, was practically beyond the reach of any concentrating machine at that time known, and could therefore go to tailing. A Dorr thickener[2] 130 ft. in diameter, the first one of such large size, was devised to recover the water from this overflow before it was allowed to go to waste. (See Fig. 3.)

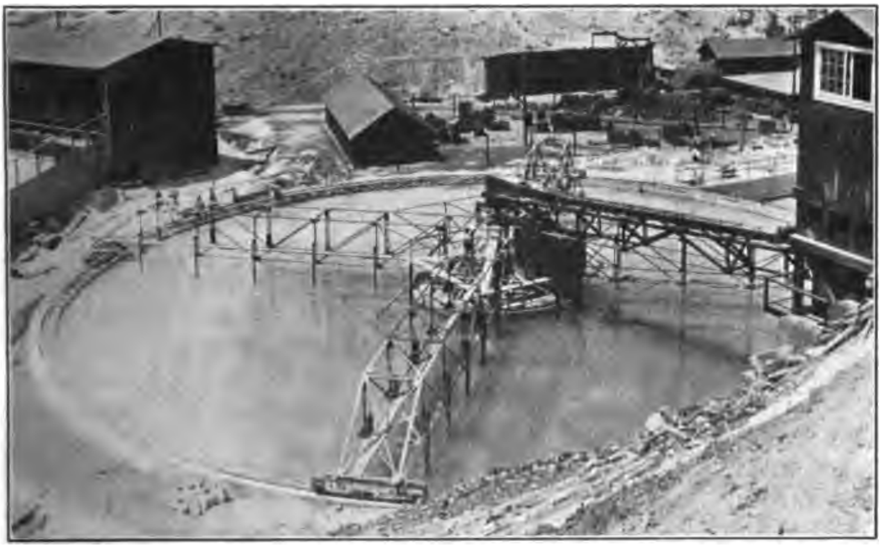

FIG. 3.—DORR THICKENER OF 130-FT. DIAMETER AT ARIZONA COPPER CO.'S PLANT.

By this plan, the treatable portion of the slime could be handled by the complement of vanners already installed in the company's mill. This arrangement, in conjunction with the saving in floor space, resulting from the introduction of the Butchart riffle, made it easily possible to double the capacity of the plant under practically the original roof. While it did not promise to recover a larger percentage than usual of the truly slimed copper, it did promise to give the best possible results in one-half the space.

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[2] Described in *Engineering and Mining Journal*, Vol. 100, No. 4, p. 131 (July 24, 1915).

By the time this flow sheet had been put into practical operation, the Inspiration experiments were attracting widespread attention and flotation was beginning to be taken seriously as a concentration agent for the handling of copper ores. It was, however, still regarded as an auxiliary process and was thought to be inapplicable to ores carrying an excess of talc or clay like the Clifton-Morenci ores. Some experiments with the Elmore process on these ores in former years had been unsuccessful and laboratory work which we did later on a very small scale with the meager information available seemed to corroborate this theory. Space was reserved, however, in a proper place in the mill building, for the installation of flotation equipment in case further development should prove its adaptability—at least for some appreciable portion of the tonnage.

Meanwhile, the Inspiration company had built the 600-ton "pilot" mill in which the new process was rapidly graduating from an auxiliary into the main method of separation, in the manner so fully and interestingly described by Dr. Gahl.[3] Starting with semi-mysterious compounds, Inspiration had soon found that simple flotation reagents were equally efficacious. Through the kindness of Mr. Mills, I secured a drum of cresylic acid and some pine oil with which to try a few experiments at Morenci.

The tailing from the No. 6 Concentrator was at that time discharged into Morenci Canyon and cascaded along for about 1 mile before being taken into a flume to be carried to the impounding dams. The creek bed was rough and steep, inducing great agitation of the pulp and resulting in the production of large amounts of white froth which floated down the stream. This froth carried no concentrations of copper minerals, but I thought it might be possible to change its character and produce a mineral froth by the use of flotation reagents introduced where the tailing left the mill, and thus possibly secure from the natural situation afforded some benefit at little cost.

A small can of the cresylic acid was arranged to drip into the tailings launder at a point where the tailings made the initial plunge into the creek bed. The results were instantaneous and very gratifying. Black froth began to collect in eddies and float downstream for a few yards to a second plunge where we were greatly surprised to find that it became white again on account of the instant dropping of the metallic load. Feeding the reagent into the stream immediately above the second plunge would not cause a mineral froth to rise as in the first plunge, and the reason was finally located as being the effect of a town sewer which was discharging under the surface into the creek between the two pools; the sewage effectively killed the metal-carrying capacity of the froth.

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[3] *History of the Flotation Process at Inspiration*, in this *Bulletin*.

Cresylic acid was then added to the feed of a regrinding Hardinge mill which was discharging into a long drag-belt classifier. The results were again most encouraging; black mineral froth began immediately to appear and to collect in large volume upon the relatively still water in the drag-belt trough. This rough froth concentrate was found to assay over 45 per cent. copper; the product contained 35 per cent. insolubles, mostly in the form of coarse sand, mechanically suspended in the froth and easily separated by screening. It was found that 1 per cent. of the copper in the froth concentrate was in oxidized form; 22 per cent. of the concentrate was too coarse to pass a 100-mesh screen, and this portion carried only 0.87 per cent. copper, while the minus 100-mesh material carried 46 per cent. copper, 1.32 per cent. of which was oxidized, and but 16 per cent. of insolubles. The high grade of the froth concentrate was astonishing, showing that chalcocite and bornite predominated in it.

A few days later I made a bank of tube grates, consisting of six parallel 1-in. pipes made up with return bends. The pipes were drilled full of small holes and were wrapped with cotton blanket tied with spirally wound wire. This tube-grate air filter was put into the drag-belt trough as deeply as possible, without touching the belt. Coarse sand could pass through between the grates and be removed by the belt underneath. The pipes were supplied with compressed air for the purpose of creating additional froth, and it was found that the product made, without further treatment, assayed 40 per cent. copper, 1.14 per cent. of which was oxidized, and that it carried but 20.4 per cent. insolubles.

Plans for a small frothing machine of the mechanical-agitation type were immediately made, and on July 20 the apparatus was tried with the colloid separator fines as feed, with the following remarkable result: Feed, 2.32 per cent. copper, of which 0.62 per cent. was oxidized; the concentrates produced assayed 20.4 per cent. copper, of which 1.18 per cent. was oxidized; the tailing carried 0.52 per cent. copper, of which 0.38 per cent. was oxidized, leaving only 0.14 per cent. sulphide copper as the rejection of the machine. This was an extraction of more than 79 per cent. of the total copper, and more than 92 per cent. of the available (sulphide) copper. This showed clearly that much could be expected in the application of the new process to Morenci ores.

The Cananea Consolidated Copper Co., in Sonora, Mexico, had been experimenting with the use of some Flinn-Towne pneumatic flotation units in its concentrating department. The plant had been temporarily shut down on account of revolutionary troubles, and arrangements were made by Dr. Ricketts for the removal of one of these units to the No. 6 Concentrator at Morenci. The apparatus was installed under the direction of the Flinn-Towne people in the space reserved for flotation and was operated for several weeks with very gratifying results as to recoveries. These experiments demonstrated clearly that the flotation process would be suited to the recovery of slimed copper sulphides in the Morenci ores. But the Flinn-Towne units were not of a size suitable for use in the equipment of a large plant, or for the handling of large tonnages, except by using a great number of them. It was thought that their capacity could not be enlarged to advantage because of the difficulty with the air-emitting medium used, which was in circular-disk form with central discharge. These disks could not be made larger in diameter without increasing the difficulty coming from "blinding" of the air-emitting surfaces, through the lodging of coarse particles upon them, and from the formation of vortices by the larger volumes discharged through the single opening in the center which entrained froth with the reject.

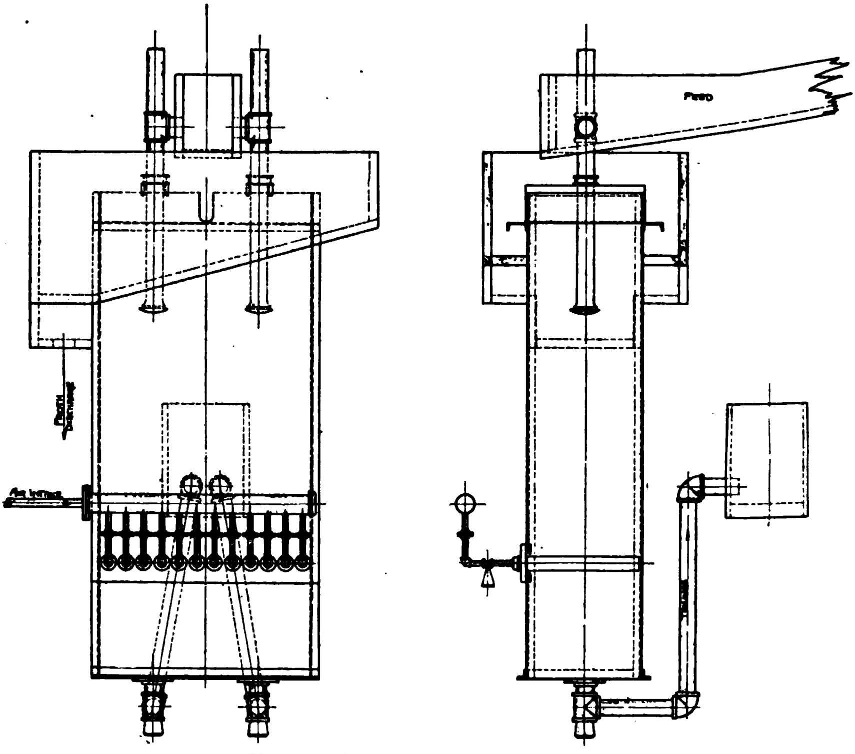

FIG. 4.—SIMPLE TUBE-GRATE CELL.

The tube-grate idea previously tried in the drag-belt tank seemed to be a better way to admit the air because nothing could lodge upon the air-emitting elements to blind them, and constriction of the passage for the pulp and water would be avoided. This tube-grate idea therefore formed a basis on which to design units of large capacity for practical mill work and especially to solve the limited-space problem at No. 6 Concentrator. Accordingly, a tube-grate cell was installed, in January, 1915. This simple cell is shown in Fig. 4. It was used for some time to demonstrate the tube-grate idea and served as a "cleaner" in the subsequent work done with a full-sized machine.

The demonstration of the new tube-grate cell was such a success that a three-stage machine, to have a capacity of 400 tons per day, called the "C-B" machine, was designed and made. Another one of the same kind and size was made concurrently for the Inspiration Consolidated Copper Co., and both of them were started in operation early in March. The Inspiration machine, which is illustrated in Dr. Gahl's paper, gave good results, proving the design to be substantially correct. The Morenci machine was working in corrosive water, which resulted in the formation of rust on the steel tubes and the gradual closing of the openings. The air supply was found to be contaminated with grease and oil from the blower bearings, entrained muddy water in the air, etc., which closed the pores of the air filter from the inside. Some delay was experienced in overcoming these difficulties. That they were of minor importance was proved by the almost uninterrupted success and approximately perfect operation of the duplicate machine operating concurrently at Inspiration.

The C-B machine at Morenci made large volumes of very rich froth and had become immediately profitable by reason of its being able to handle a large tonnage and save copper which would be otherwise beyond the reach of concentration. It was therefore kept in operation as a profit maker, even though working under the handicap of partially clogged tubes, blower troubles, etc. The daily tonnage handled during the month of April, 1915, was from 125 to 390 tons per day, with an average of 209; the recovery made by the machine was from 35 to 79 per cent. of the sulphide copper present in the feed, with an average for the period of 65 per cent.

The blower used was an old one borrowed from the mining department, where it had been used in ventilation. It was designed for not more than 3½ lb. pressure, and the developing of 6 lb. in it deflected the shafts and caused the impellers to rub upon the sides of the machine, which had to be water-jacketed to keep down the heat developed. It was much larger than necessary, and a great excess of air was blown off from open valves. Therefore, no record of the amount of air used or power required, could be even approximately obtained.

In spite of these minor difficulties it was proved: That flotation could be applied to these ores with great advantage; that the copper in the mill tailing could be reduced to 0.50 per cent. (of which 0.25 to 0.30 per cent. was oxidized and beyond the reach even of flotation); that this result could be improved by finer grinding in the Hardinge mills; that the

TABLE 1.—*Detailed Results of Tests on C-B and Callow Flotation Machines for Month of July*, 1915.

SCREEN ANALYSIS C-B MACHINE

Mesh	Machine Tails							Machine Concentrates							Vanner Tails						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total
+ 20	......	......	.....	....	......	......	......	......	......	.....	....	....	......	......	......	......	.....	....	......	......	...
+ 28	1.1	1.1	0.77	....	0.85	1.41	1.41	......	......	.....	....	....	......	......	1.4	1.4	0.66	....	0.92	2.11	2.11
+ 35	3.8	4.9	0.65	....	2.47	5.51	5.51	......	......	.....	....	....	......	......	4.4	5.8	0.60	....	2.64	6.04	8.15
+ 48	8.0	12.9	0.79	....	6.32	10.49	16.00	0.5	0.5	2.22	....	1.11	0.04	0.04	9.9	15.7	0.60	....	5.94	13.61	21.76
+ 65	9.9	22.8	0.80	....	7.92	13.14	29.14	0.6	1.1	14.72	....	8.83	0.32	0.36	11.0	26.7	0.51	....	5.61	12.85	34.61
+100	14.7	37.5	0.87	....	12.79	21.23	50.37	1.6	2.7	15.82	....	25.31	0.90	1.26	16.6	43.3	0.48	....	7.97	18.25	52.86
+150	11.2	48.7	0.79	....	8.85	14.69	65.06	7.9	10.6	20.16	....	159.26	5.64	6.95	10.9	54.2	0.37	....	4.03	9.23	62.09
+200	6.9	55.6	0.67	....	4.62	7.67	72.73	10.6	21.2	23.99	....	254.29	9.08	16.03	6.3	60.5	0.37	....	2.33	5.34	67.43
−200	44.4	100.0	0.37	....	16.43	27.27	100.00	78.8	100.00	29.83	....	2,350.60	83.97	100.00	39.5	100.0	0.36	....	14.22	32.57	100.00
Total...	60.0	......	.....	....	60.25	100.00	......	100.00	......	.....	....	2,799.40	100.00	......	100.0	......	.....	....	43.66	100.00	......
Check..	......	......	0.61	0.24	......	......	......	......	......	27.84	1.44	Fe 19.8	Ins. 18.8	......	......	......	0.41	0.25	......	......	......

Mesh	Vanner Concentrates							Table Tails							Table Concentrates						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu.	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total
+ 20	0.2	0.2	2.02	....	0.40	0.04	0.04	......	......	.....	....	....	......	......	0.7	0.7	1.53	....	1.07	0.11	0.11
+ 28	0.6	0.8	3.63	....	2.18	0.24	0.28	0.9	0.9	0.65	....	0.59	1.26	1.26	1.0	1.7	4.23	....	4.23	0.45	0.56
+ 35	1.6	2.4	7.26	....	11.64	1.28	1.56	3.1	4.0	0.66	....	2.05	4.36	5.62	2.0	3.7	7.75	....	15.50	1.66	2.22
+ 48	4.7	7.1	9.08	....	42.68	4.70	6.26	7.2	11.7	0.60	....	4.32	9.20	14.82	6.3	10.0	9.79	....	61 68	6.59	8.81
+ 65	8.2	15.3	10.90	....	89.38	9.84	16.10	9.8	21.0	0.58	....	5.68	12.09	26.91	14.3	24.3	10.20	....	145.86	15.58	24.39
+100	17.4	32.7	11.60	....	201.84	22.23	38.33	14.8	35.8	0.54	....	7.99	17.01	43.92	36.8	61.1	8.77	....	322.74	34.48	58.87
+150	17.6	50.3	11.60	....	204.16	22.49	60.82	11.7	47.5	0.44	....	5.15	10.96	54.88	24.0	85.1	9.79	....	234.96	25.10	83.97
+200	13.4	63.7	8.26	....	110.68	12.19	73.01	7.1	54.6	0.49	....	3.48	7.41	62.29	8.9	94.0	10.81	....	96.21	10.28	94.25
−200	36.3	100.0	6.75	....	245.02	26.99	100.0	45.4	100.0	0.39	....	17.71	37.71	100.0	6.0	100.0	8.97	....	53.82	5.75	100.00
Total...	100.0	......	.....	....	907.96	100.0	......	100.0	......	.....	....	46.97	100.0	......	100.0	......	.....	....	936.07	100.0	......
Check .	......	......	7.96	1.91	......	......	......	......	......	0.41	0.24	....	......	......	......	......	8.96	1.40	Ins. 43.6	......	......

SCREEN ANALYSIS CALLOW MACHINE

Mesh	Machine Tails							Machine Concentrates							Vanner Tails						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu.	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu.	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. Total	Cum. Per Cent of Total
+ 20	......	......	......	....	......	......	......	......	......	......	....	......	......	......	......	......	......	....	......	......	......
+ 28	0.8	0.8	0.94	....	0.75	1.26	1.26	......	......	......	....	......	......	......	0.4	0.4	0.61	....	0.24	0.61	0.61
+ 35	1.2	2.0	0.81	....	0.97	1.63	2.89	......	......	......	....	......	......	......	1.8	2.2	0.53	....	0.95	2.40	3.01
+ 48	5.9	7.9	0.81	....	4.78	8.03	10.92	......	......	......	....	......	......	......	7.1	9.3	0.49	....	3.48	8.81	11.82
+ 65	10.4	18.3	0.87	....	9.05	15.21	26.13	0.6	0.6	5.14	....	3.08	0.48	0.48	11.4	20.7	0.49	....	5.59	14.15	25.97
+100	16.8	35.1	0.87	....	14.62	24.57	50.70	1.7	2.3	13.20	....	22.44	3.50	3.98	17.7	38.4	0.54	....	9.56	24.20	50.17
+150	12.5	47.6	0.80	....	10.00	16.80	67.50	8.8	11.1	19.15	....	168.52	26.25	30.23	12.1	50.5	0.44	....	5.32	13.47	63.64
+200	7.8	55.4	0.65	....	5.07	8.52	76.02	10.1	21.2	24.29	....	245.33	38.22	68.45	7.1	57.6	0.29	....	2.06	5.22	68.86
−200	44.6	100.0	0.32	....	14.27	23.98	100.0	78.8	100.0	25.70	....	2,025.16	31.55	100.00	42.4	100.0	0.29	....	12.30	31.14	100.0
Total...	100.00	......	......	....	59.51	100.0	......	100.0	......	......	....	2,464.53	100.0	......	100.0	......	......	....	39.50	100.0	......
Check..	......	......	0.66	0.23	......	......	......	......	......	24.80	1.24	Fe 19.4	Insol. 24.6	......	......	......	0.37	0.15	......	......	......

Mesh	Vanner Concentrates							Table Tails							Table Concentrates						
	Solids		Assay			Copper		Solids		Assay			Copper		Solids		Assay			Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum Per Cent. of Total	Per Cent. Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cn	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Per Cent. of Total	Cum. Per Cent of Total
+ 20	0.4	0.4	2.42	....	0.97	0.15	0.15														
+ 28	0.6	1.0	1.81	....	1.09	0.17	0.32	0.5	0.5	0.88	....	0.44	1.10	1.10	0.8	0.8	3.67	....	2.94	0.34	0.34
+ 35	1.4	2.4	3.12	....	4.37	0.66	0.98	1.3	1.8	0.64	....	0.83	2.08	3.18	0.5	1.3	5.91	....	2.96	0.35	0.69
+ 48	3.3	5.7	6.35	....	20.96	3.18	4.16	4.2	6.0	0.57	....	2.39	5.98	9.16	5.2	6.5	9.59	....	49.87	5.82	6.51
+ 65	6.0	11.7	10.40	....	67.40	9.48	13.64	10.4	16.4	0.47	....	4.89	12.25	21.41	17.7	24.2	9.59	....	169.74	19.80	26.31
+100	16.0	27.7	12.81	....	204.96	31.13	44.77	16.0	32.4	0.44	....	7.04	17.63	39.04	40.0	64.2	8.06	....	322.40	37.61	63.92
+150	15.0	42.7	12.10	....	181.50	27.57	72.34	11.9	44.3	0.47	....	5.59	14.00	53.04	21.8	86.0	8.57	....	186.83	21.80	85.72
+200	12.1	54.8	6.76	....	81.80	12.42	84.76	7.8	52.1	0.44	....	3.43	8.59	61.63	8.2	94.2	8.87	....	72.73	8.48	94.20
−200	45.2	100.0	2.22	....	100.34	15.24	100.0	47.9	100.0	0.32	....	15.32	38.37	100.0	5.8	100.0	8.57	....	49.71	5.80	100.0
Total...	100.0	......	......	....	658.39	100.0	......	100.0.	......	....	....	39.93	100.0	......	100.0	......	......	....	857.18	100.0	......
Check..	......	......	6.40	2.13	......	......	......	......	......	0.45	0.25	......	......	......	......	......	7.84	1.44	Fe· 21.0	......	......

TABLE 2.—*Comparative Results Obtained in Operatian of C-B and Callow Flotation Machines at Concentrator No. 6, Arizona Copper Company, Ltd., Morenci, Ariz.*

May, 1915

		C-B Per Cent.	Callow, Per Cent.
Flotation machine tails	Total copper	0.72	0.81
	Oxidized copper	0.38	0.30
	Sulphide copper	0.34	0.51
Flotation machine concentrates	Total copper	38.19	24.82
	Insolubles	25.00	27.00
Vanner tails	Total copper	0.53	0.46
	Oxidized copper	*	0.18
	Sulphide copper	*	0.28
Vanner concentrates	Total copper	8.52	8.89
	Insolubles	*	30.60

June, 1915

		C-B Per Cent.	Callow, Per Cent.
Flotation machine tails	Total copper	0.71	0.69
	Oxidized copper	0.30	0.28
	Sulphide copper	0.41	0.41
Flotation machine concentrates	Total copper	35.24	25.41
	Insolubles	23.80	26.00
Vanner tails	Total copper	0.41	0.42
	Oxidized copper	0.23	0.22
	Sulphide copper	0.18	0.20
Vanner concentrates	Total copper	9.35	9.51
	Insolubles	*	* ....

July, 1915

		C-B, Per Cent.	Callow
Flotation machine tails	Total copper	0.61	0.66
	Oxidized copper	0.24	0.23
	Sulphide copper	0.37	0.43
Flotation machine concentrates	Total copper	27.84	24.80
	Insolubles	18.80	24.60
Vanner tails	Total copper	0.41	0.37
	Oxidized copper	0.25	0.15
	Sulphide copper	0.16	0.22
Vanner concentrates	Total copper	7.96	6.40
	Insolubles	*	* ....
Table tails	Total copper	0.41	0.45
	Oxidized copper	0.24	0.25
	Sulphide copper	0.17	0.20
Table concentrates	Total copper	8.96	7.84
	Insolubles	43.60	* ....

* No assay.

	C-B	Callow
Daily tonnage rate, average for the month of July, 1915	479	139

SEE NOTE, p. 1621.

same simple reagents used elsewhere would apply. Further, a new type of pneumatic flotation machine, well adapted to the conditions at No. 6 Concentrator, had been successfully developed and its performance demonstrated on a full-sized unit. The new type machine would not be in any way embarrassed by the oversize coming from Hardinge mills.

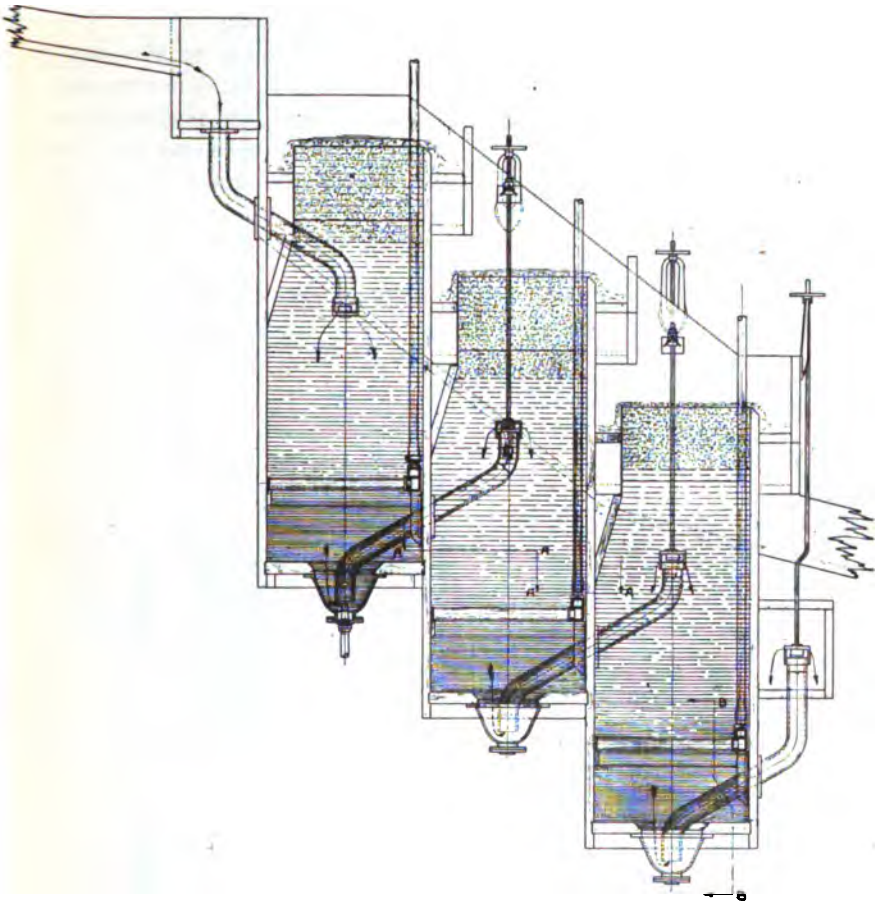

FIG. 5.—LONGITUDINAL SECTION OF C-B FLOTATION MACHINE. THREE CELLS IN SERIES.

All of the drag-belt overflow being handled by the colloid separator could go to a few of these new machines where all of the rich slimed sulphide copper would be taken out. Or the whole tonnage of reground

---

NOTE TO TABLE 2.—Without protection against "blinding" of the air-emitting media by sand or "oversize" the C-B machine handled considerably more than three times the tonnage handled by the Callow in July, and did this without detriment to the metallurgical work.

material produced in the Hardinge mills could go directly to the new type frother in which the slimed copper sulphides would be removed. The thoroughly frothed sands could then be treated on tables and vanners for the removal of the remaining sulphide particles which were too coarse to be separated by flotation. Since there would then be no embarassing losses in the slime part of the feed, these machines would work efficiently, and a maximum mill recovery would result.

After it became evident that flotation would apply to Morenci ores and before the value of the tube-grate idea was fully demonstrated, it was decided to install and experiment with a standard Callow flotation unit of 200 tons capacity, consisting of four rougher cells and one cleaner. This equipment was not received until after the full-sized C-B unit of 400 tons daily capacity had been installed.

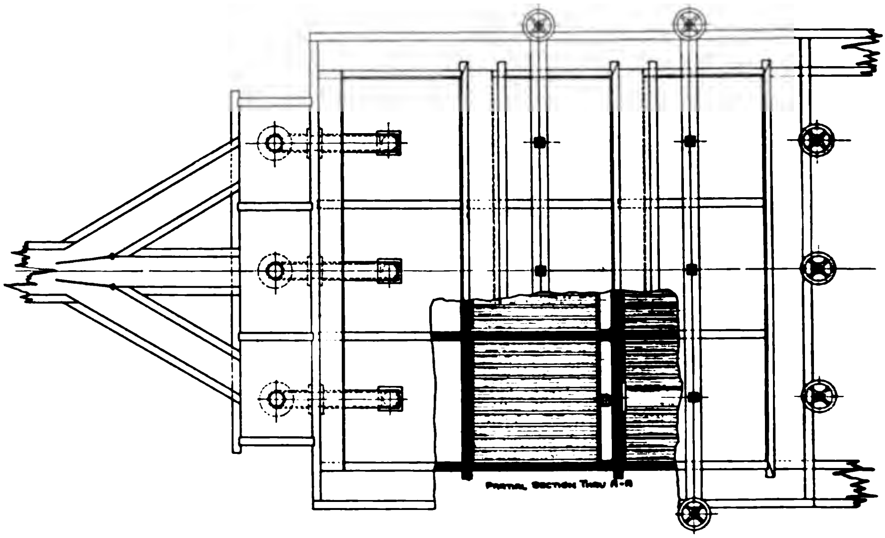

FIG. 6.—PLAN OF C-B FLOTATION MACHINE.

The Callow equipment was started in operation May 24, 1915, and competitive operation was carried on for about 3 months. The recoveries proved to be very much alike although the feed was not identical. The Callow apparatus is not adapted to handle coarse particles of feed or oversize, and had to be protected by a screen or spitzkasten. It will handle about one-half the normal tonnage of the C-B machine, occupying the same mill space. A summary of the results obtained for the months of May, June, and July, 1915, also details showing the work for the month of July, are given in Tables 1 and 2. The performance is shown to be substantially parallel as to quality of work done, but quite different as to quantity of tonnage handled.

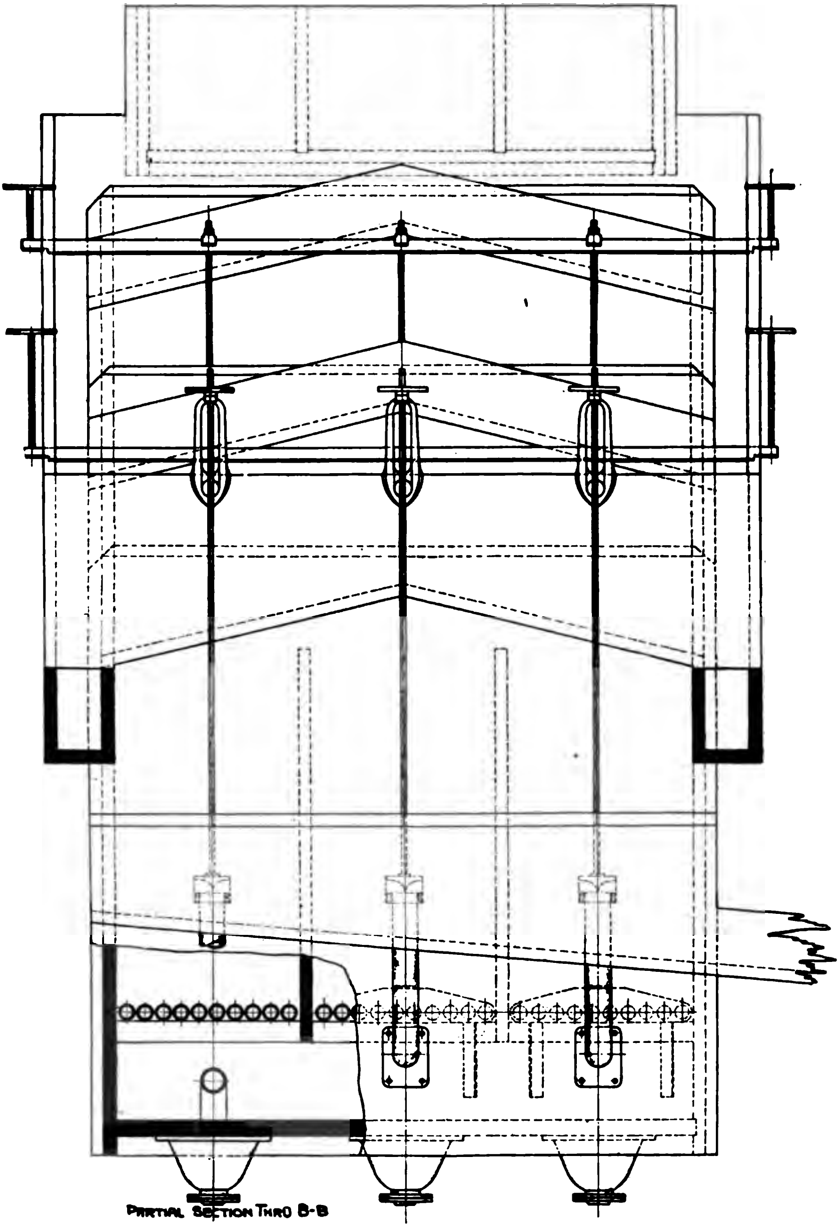

FIG. 7.—FRONT ELEVATION OF C-B FLOTATION MACHINE.

As mentioned before, the air and power consumption was not determined in the C-B installation because there was no way to take even approximately correct measurements. But the experience at Inspiration, where the C-B and Callow systems were also being operated in parallel, showed that substantially the same amounts of air and power were used by each system.

Experience suggested wider launders for froth, larger cleaner capacity, and simplified tube-grate construction of the C-B unit. These ideas are incorporated in the new design shown in Figs. 5, 6 and 7 in which it will be noted that the machine is merely a stationary wooden box suitably arranged to receive air-emitting tubes which are dropped in from the top and rest upon ledges at the proper level in the pulp to be treated. These air-emitting elements are connected to the air supply by the use of rubber hose. They can be taken out or put in without cutting off the feed or shutting down the machine, in case this should be of advantage. An air pressure of 5 lb. is required. The air-emitting surface in the C-B machine is more than twice the complete cross-sectional area of the frothing compartments, and even if a ridge of sand should lodge upon the extreme top of the tubes and partially cut off the air supply, the remaining unobstructed area would still be larger than the whole crcss-section.

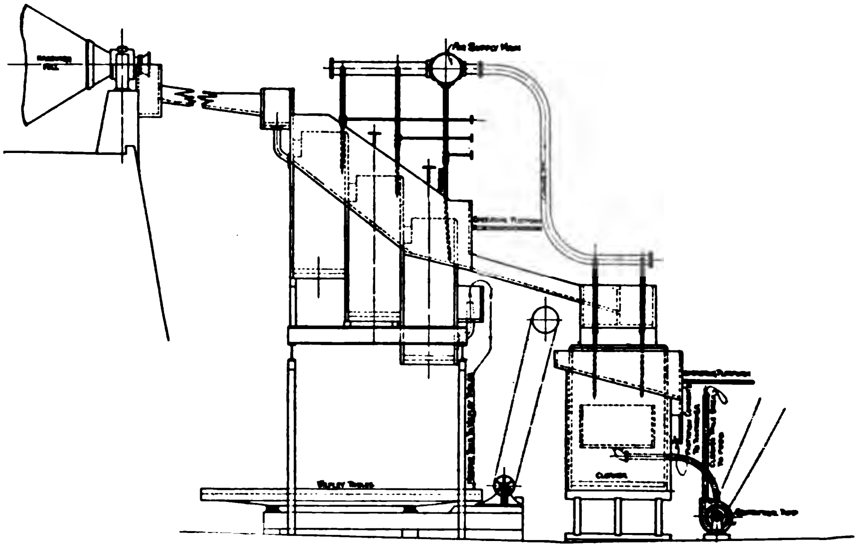

FIG. 8.—ARRANGEMENT AT MILL OF CANANEA CONSOLIDATED COPPER CO., CANANEA, MEX., USING THE C-B 3-STAGE FLOTATION MACHINE.

Fig. 8 shows the improved form with cleaners as arranged in the mill of the Cananea Consolidated Copper Co., and is typical of the arrangement adopted for the later models. There are eight units in this installation. They are operated under conditions varying greatly as to quality

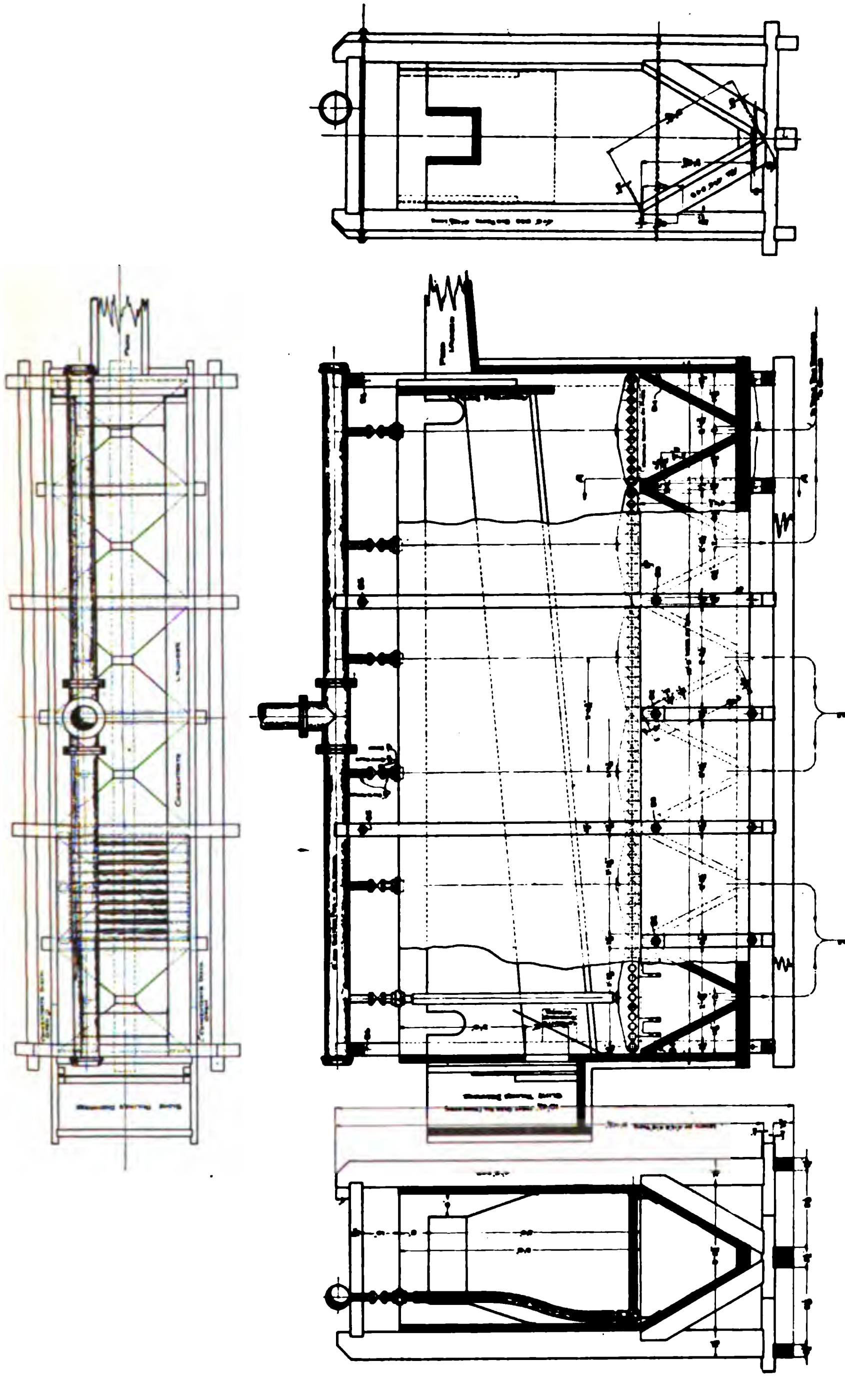

FIG. 9.—SPITZKASTEN FROTHER EQUIPPED WITH TUBE-GRATE AIR FILTER.

of ores, there being a wide range of iron and copper sulphide conditions and all conditions of oxidation in the ore handled. The operation of the Cananea machines has been repeatedly interrupted by the internal strife in Mexico and no deductions of value are at present available.

Fig. 9 shows an application of the tube-grate idea to a spitzkasten type of frother. One of these machines was made and installed in No. 6 Concentrator at Morenci but was taken out before it was tried. It seems to embody advantages of much promise in a frothing first flow-sheet and will soon have a trial to determine its value in the simplified concentration of ore that is amenable to flotation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

History of the Flotation Process at Inspiration

BY RUDOLF GAHL,[*] PH. D., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

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THE history of flotation in America is very short, at least as far as the large-scale application of the process is concerned. It is remarkable how many important developments have taken place in the last few years and are already being extensively utilized. What was new a year ago, seems almost commonplace now. For this reason it is with hesitation that I follow the suggestion of Dr. Ricketts, President of the Institute, to describe the experiences of the Inspiration Consolidated Copper Co. with the flotation process.

TESTS CONDUCTED IN SMALL TEST MILL

When the Inspiration company first decided to build a concentrating plant, nothing was known about flotation, and the process was to be gravity concentration pure and simple.

Demonstration Tests Conducted by Minerals Separation Co.

While plans were being prepared by H. Kenyon Burch (who had been intrusted with the design and construction of the concentrator) the Minerals Separation Co., a concern at that time little known in America, asked and obtained permission to demonstrate the value of its flotation process for Inspiration ore. As a consequence, a small 50-ton flotation machine of standard design was added to the company's test plant and started to operate in the beginning of 1913. This marks the beginning of flotation at Inspiration.

The results obtained with this machine, which was operated by members of the Minerals Separation staff, so far surpassed what this company anticipated that it was decided to continue flotation tests for this purpose, and two of the flotation experts of the Minerals Separation Co., I. L. Greninger and G. A. Chapman, were retained. L. R. Wallace, now superintendent of the Miami works of the International Smelting Co., was at that time metallurgist of the Inspiration company and in that capacity took an active part in these tests. Great credit is due to him for his quick recognition of the possibilities of flotation.

Flotation Tests Conducted by Inspiration Co.

The tests led to the conclusion that it would be advisable to incorporate flotation into the concentrating process. Doubt existed only as to the extent to which this should be done. The first design brought out by Mr. Burch only called for flotation treatment of the concentrator tailings. While the tests were progressing, it became more and more evident, however, that flotation should form a more essential part of the milling process, and it was finally decided to leave out all the complications which are usually adopted in gravity concentration plants for the purpose of recovering as much as possible of the mineral values of the fines, and to rely on flotation alone for this purpose.

Sampling of Orebody

This decision was reached only after it had been established by numerous tests that the orebody as a whole was suited for flotation treatment. It was found, it is true, that a portion of the ore in the mine was unsuitable for flotation by the process in question, but the amount of this ore was established to be only a small fraction of the total. Besides, the tests proved that the ore contained in this fraction lost its refractory character when mixed with the rest of the ore.

Mr. Greninger gives the following description of the manner in which these tests were carried out:

"Sample lots, each amounting to about 10 tons and representing from 30 to 35 ft., were blasted from the back and sides of the drifts and after suitable crushing and grinding treated in the small flotation machine mentioned above.

"The results were very erratic, some showing good recovery with a high grade of concentrate, while others returned concentrate of a very low grade and also showed a low extraction.

"After a considerable number of tests had been made, it was found that the good results were always obtained when treating ore of a schist gangue, while poor results resulted when the gangue was the altered and kaolinized granite which forms a part of the Joe Bush orebody.

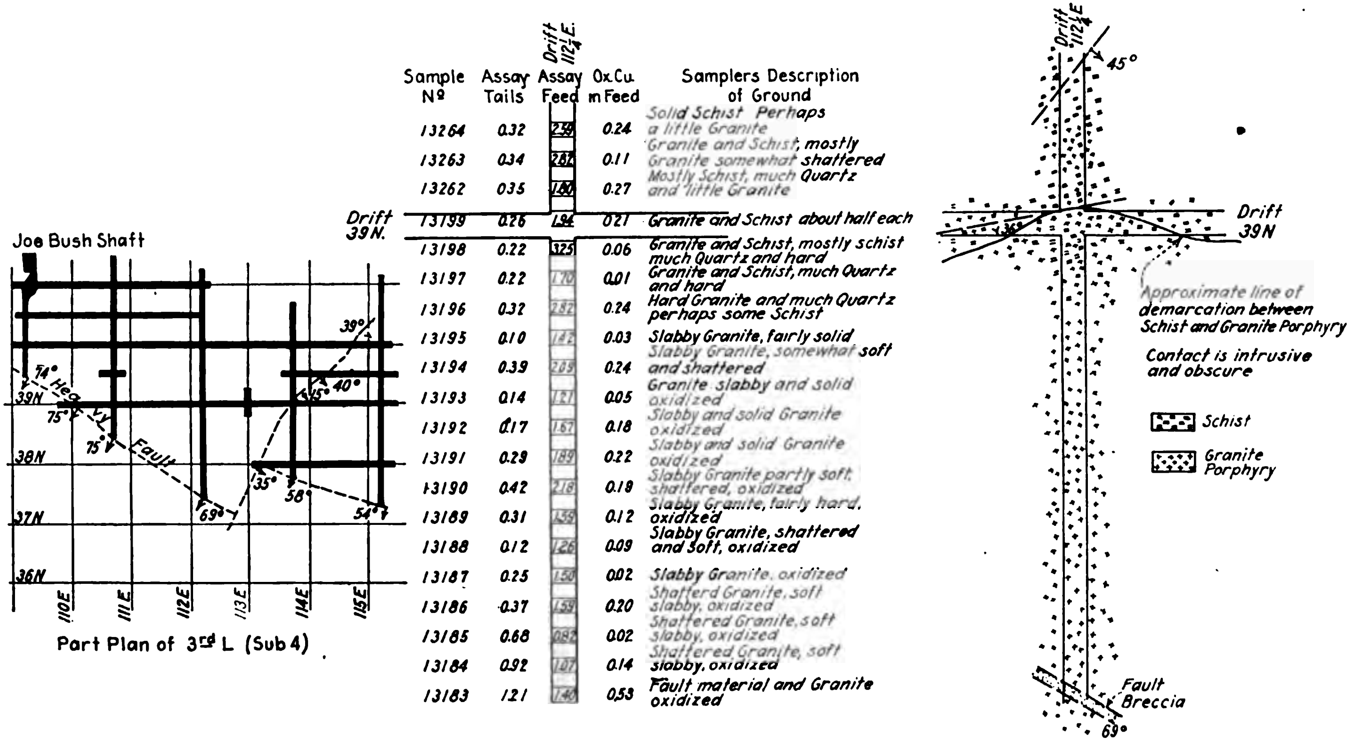
"Ten-ton samples were taken from various parts of the mine for the purpose of determining the extent and amount of this granite ore. These lots were taken from the drifts crossing the orebody from south to north, starting in the granite ore and continuing along each drift until the schist gangue was encountered.

"In all of these tests the results were uniformly good while treating the schist ore and poor while treating the granite ore.

"Subsequently, another series was inaugurated with the view of determining the amount of granite ore that could be mixed with the schist ore without interfering with the treatment of this mixed ore by flotation.

"Various percentages of granite ore were mixed with clean schist and the mixture treated by flotation. It was found that 10 per cent. of granite caused no change in the behavior of the ore in the flotation plant and when 20 per cent. of the granite was mixed with the schist ore only a slight difference was noted, this difference consisting in a slight lowering of the grade of the concentrate with a corresponding falling off in extraction."

The conditions described by Mr. Greninger are well illustrated by a drawing made by C. E. Arnold of the mine-engineering staff of the com-



Sample Nº	Assay Tails	Assay Feed	Ox.Cu. in Feed	Samplers Description of Ground
13264	0.32	2.59	0.24	Solid Schist Perhaps a little Granite
13263	0.34	2.82	0.11	Granite and Schist; mostly Granite somewhat shattered
13262	0.35	1.80	0.27	Mostly Schist, much Quartz and little Granite
13199	0.26	1.94	0.21	Granite and Schist about half each
13198	0.22	3.25	0.06	Granite and Schist, mostly schist much Quartz and hard
13197	0.22	1.70	0.01	Granite and Schist, much Quartz and hard
13196	0.32	2.82	0.24	Hard Granite and much Quartz perhaps some Schist
13195	0.10	1.42	0.03	Slabby Granite, fairly solid
13194	0.39	2.05	0.24	Slabby Granite, somewhat soft and shattered
13193	0.14	1.21	0.05	Granite slabby and solid oxidized
13192	0.17	1.67	0.18	Slabby and solid Granite oxidized
13191	0.29	1.89	0.22	Slabby and solid Granite oxidized
13190	0.42	2.18	0.18	Slabby Granite partly soft, shattered, oxidized
13189	0.31	1.59	0.12	Slabby Granite, fairly hard, oxidized
13188	0.12	1.26	0.09	Slabby Granite, shattered and soft, oxidized
13187	0.25	1.50	0.02	Slabby Granite, oxidized
13186	0.37	1.59	0.20	Shatterd Granite, soft slabby, oxidized
13185	0.68	0.82	0.02	Shattered Granite, soft slabby, oxidized
13184	0.92	1.07	0.14	Shattered Granite, soft slabby, oxidized
13183	1.21	1.40	0.53	Fault material and Granite oxidized

FIG. 1.—MAP SHOWING GEOLOGY AND RESULTS FROM LABORATORY FLOTATION TESTS ON 5-FT. SAMPLES TAKEN AT 10-FT. INTERVALS ALONG DRIFT BETWEEN 3740 AND 3935 N.

pany, which represents laboratory flotation results on samples taken from a drift in the Joe Bush orebody (Fig. 1). The results show clearly that the granite by itself does not interfere with flotation, but only the fault material, evidently corresponding to what Mr. Greninger terms kaolinized granite.

Other Results Obtained in Small Test Mill

The test conducted in the small test plant established in a general way the physical conditions under which Inspiration ores could be treated advantageously; for instance, it was decided that raising the temperature did not improve the results obtained in proportion to the extra expense of such procedure.

As far as flotation oils are concerned, those in charge of the tests came to the conclusion that cresylic acid (98 per cent. pure) should be used as the main flotation agent, and should be supplemented by crude turpentine. As the most important result brought out by these tests, I consider the discovery that the flotation agents may profitably be added in the grinding machines, while it had formerly been the customary practice to add the "oils" in agitating tanks especially provided for the purpose. This discovery had an important bearing on the later developments in the Inspiration milling practice, inasmuch as it paved the way for the use of much heavier oils; for instance, coal tar which it is impossible to amalgamate thoroughly with the pulp in agitating tanks. Mr. Chapman, I think, made this important discovery during this period (U. S. Patent 1,102,874).

The details of the flow sheet to be followed were left to be decided by tests on a larger scale. Only this much was settled: That no concentration of any kind, either flotation or gravity, should be attempted before the ore was reduced to the fineness required for the flotation process. This decision was brought about by the fact that excellent recoveries were obtained when this procedure was followed, and was, of course, also strongly urged by considerations of simplicity in the milling process and cheapness of milling operations.

In these tests a 50-ton flotation machine of Minerals Separation Co. Standard type was used. In the general design, it was similar to the larger machine later put in use. It had eight agitating compartments with propellers of 12-in. diameter revolving with a peripheral speed of from 1,200 to 1,400 ft. per minute.

TESTS IN LARGE TEST MILL

Flow Sheet

In order to test on a large scale the points already settled in the small test mill, and to decide the points left undecided by the small-scale experiments, a 600-ton test plant was built and put into operation at the beginning of January, 1914. It was my privilege to conduct those tests in coöperation with Mr. Greninger, who represented the Minerals Separation Co., and with the representatives of other concerns, who in the course of time decided to submit flotation machines to the consideration of the Inspiration company. The flow sheet of this 600-ton test mill was extremely simple. It is illustrated in Fig. 2. The ore after being crushed to the desired fineness by machinery, which was tested out at the same time, was sent to tables; from these the concentrates went to the concentrate bins, while the tailings passed on to an 8-compartment flotation machine of the Standard Minerals Separation type with 24-in. stirrers. Later a 12-compartment machine of the same type was added and operated in parallel with the 8-compartment machine. The tailings from the flotation machines were passed on to other tables, the concentrates from which, combined with the concentrates from the upper tables and the flotation machine, went to the concentrate bins, while the tailings went to waste. The flotation machine was operated in the standard manner, the feed being introduced into the first agitating compartment, passed to the first spitzkasten, drawn to the second agitating compartment, sent from there to the second spitzkasten and drawn to the third

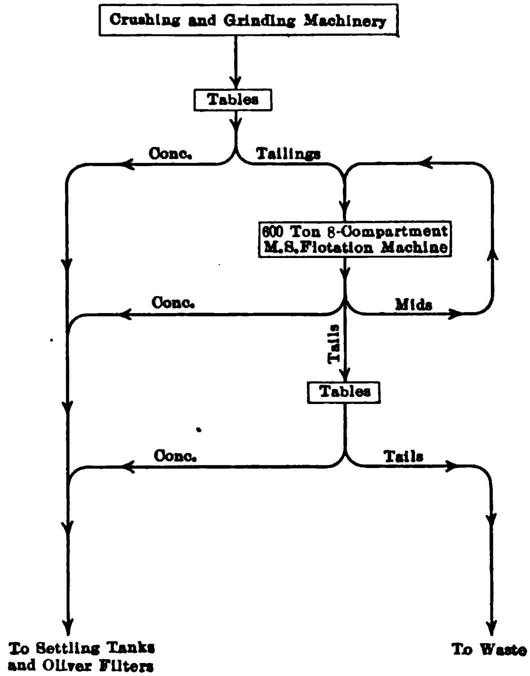

Crushing and Grinding Machinery
Tables
Conc.
Tailings
600 Ton 8-Compartment M.S. Flotation Machine
Conc.
Mids
Tails
Tables
Conc.
Tails
To Settling Tanks and Oliver Filters
To Waste

FIG. 2.—FIRST FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

agitating compartment, etc., the agitators forming the necessary suction for the transportation of the pulp from the spitzkastens to the following agitating compartments. A final concentrate was made from one or more spitzkastens, while the concentrate from those remaining was considered a middlings product and was returned to the head of the machine for retreatment.

This flow sheet was extremely simple, but after a while even this was considered too complicated, inasmuch as the necessity of table treatment ahead of flotation treatment was doubted.

Discussion of the Value of Preliminary Table Treatment

The advantages pointed out in favor of the preliminary table treatment were about as follows: Since the tables would make a certain recovery, an impoverished flotation feed would result and assist the flotation machine to make a tailings product low in copper. However, the validity of this argument was doubted for the following reasons:

In the first place, if complications are to be avoided, the preliminary table treatment has to be of a comparatively rough character; the refinements of hydraulic classification have to be dispensed with, as by its use more water would be introduced into the pulp than would be advisable for the following flotation process. It must be taken into consideration that the pulp delivered to the tables already contains about 3 tons of water to 1 ton of solid matter, experience having shown that the grinding machines, consisting of either ball or pebble mills, deliver a product of the desired fineness, about 1 or 2 per cent. on 48-mesh, when the consistency of the overflow from drag-belt classifiers, working in conjunction with the grinding mills, was carried at about this figure; experience also showed that this consistency is suitable for the flotation treatment, while greater dilution of the pulp resulted in an increased copper loss. The logical way out of this difficulty would be to introduce settling tanks, for which, however, the modern mill designer has a just abhorrence; at least, when they are to be placed in the middle of the mill. Neither the liberal use of dressing water nor a low tonnage rate would be permissible for the same reason. On this account, a high recovery could not be expected from a preliminary table treatment. During our tests it averaged around 33 per cent.

Furthermore, the assumed fact, that an impoverished feed results in a better recovery of the flotation machine, was strongly doubted by some of the flotation experts, who claimed that in order to form a froth which had the necessary carrying power for mineral, it would be better to leave as much mineral as possible in the feed to the flotation machine. In other words, their advice was to leave out the tables in order to permit the flotation machines to do more efficient work.

There is no doubt, however, that tables will catch a certain amount of mineral (especially coarser grains) which will escape in the flotation process. For this reason, tables have to be used to insure the highest recovery,[1] but it seems that tables following the flotation treatment would make up for this deficiency of the flotation machine better than tables ahead, inasmuch as they would not work under the disadvantage, known to every millman, of receiving too rich a feed. It is well known that it is possible to make a much lower table tailing working on feed containing a small percentage of copper than on a feed rich in copper. From theoretical speculations, therefore, no valid reason was advanced that increased recovery should result from table treatment ahead of the flotation treatment.

The next argument advanced by the supporters of a preliminary table treatment was that it would result in an improvement of the grade of the general concentrates, inasmuch as it would be possible to make a very clean product on the tables, thereby raising the general average of the concentrates. There is evidently some sound foundation for this point; how much, could not well be investigated without resorting to actual experiments.

A third point seemingly in favor of preliminary table treatment is this: Flotation concentrates offer more or less difficulty in mechanical handling; vacuum or pressure filters have to be resorted to for this purpose. It was pointed out, therefore, that if a certain percentage of the total concentrate could be saved on the tables, and a table concentrate produced containing only a small amount of the slime, so troublesome in filter treatment, it might be a decided advantage; the quantity of concentrate to be handled on filters might be materially reduced and economies effected in this manner. At the same time, experience shows that the flotation concentrate resulting under those conditions, because it contains less of the coarser sand, is more difficult for the filters to handle than it would be were the sand left in. Very likely, therefore, no decided improvement in the handling of the concentrates would result from the separation of the table concentrates from the flotation concentrates, at least under the conditions prevailing here.

The same objections do not hold true for tables when used after the flotation process, as in that case no limitations are imposed regarding the hydraulic and dressing water, and the tables cannot help but make an additional small recovery of concentrate. On account of the fact that the table feed is necessarily low in copper, the tailings will be correspondingly low. In all cases, therefore, where flotation treatment does not make a full recovery, or practically so, of mineral values, tabling after flotation seems imperative.[2]

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[1] This applies only to the flow sheet under discussion.

[2] This remark again refers only to the flow sheet under discussion.

Although such theoretical considerations were indulged in, no decision was based on them. To settle the main points in question, a series of tests was carried out. On alternate days, the "upper" tables were bypassed (Fig. 3) while on the remaining days, they were utilized (Fig. 2).

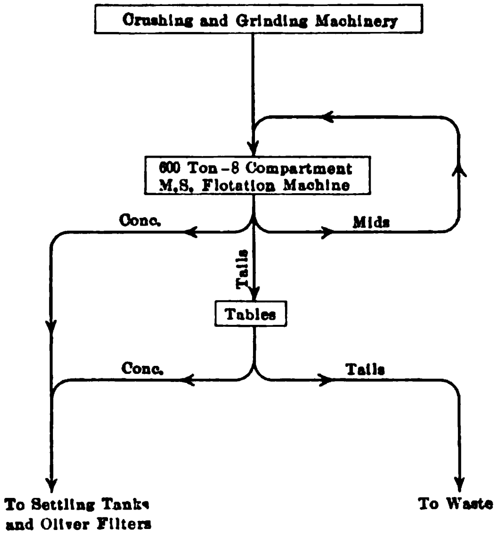
Crushing and Grinding Machinery
600 Ton-8 Compartment M.S. Flotation Machine
Conc.
Mids
Tails
Tables
Conc.
Tails
To Settling Tanks and Oliver Filters
To Waste

FIG. 3.—SECOND FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

TABLE 1.—*Comparison of Efficiency of Flow Sheet No.* 1 (*Fig.* 2) *and Flow Sheet No.* 2 (*Fig.* 3)

Description	Flow Sheet	
	No. 2	No. 1
Assay of mill feed	1.72	1.67
Assay of flotation machine feed	1.72	1.32
Assay of flotation machine tails	0.46	0.43
Assay of lower table tails	0.29	0.30
Assay of general concentrates	32.71	31.72
Recovery upper tables + flotation machine	......	75.30
Recovery flotation machine	74.30	
Additional recovery on lower tables	9.40	7.40
Total recovery	83.70	82.70
Tonnage per deck on lower tables	25.70	23.20
Tonnage per deck on upper tables	......	87.00
Oil consumption, cresol + turpentine in pounds per ton	0.95	0.82

REMARKS.—The recoveries are calculated from the feed and tailings assays and the assay of general concentrates.

CONCLUSIONS.—Roughing on upper tables reduces the tailings assay of the flotation machine slightly, while it does not seem to affect the assay of the lower tables appreciably. Increased recovery results from the use of the lower tables.

Table 1 gives the résumé, on the strength of which it was decided ot use secondary tables only and to leave the preliminary table treatment (ahead of the flotation treatment) out of the flow sheet to be adopted in the concentrator under construction. That this conclusion is justified seems to be borne out by the following facts:

In January and February, 1914, the lower table floor was not in operation. For this reason, the flotation tailings were not treated on tables. There was, however, table treatment ahead of flotation, the following results being obtained: Table recovery, 39.42 per cent.; flotation recovery, 33.35 per cent.; total recovery, 72.77 per cent. From March to August, 1914, various flow sheets were tried; tables were sometimes used preceding and sometimes following flotation, and sometimes in both places. The figures on recovery are, for this reason, not comparable with the preceding ones. From September to December, 1914, no tables were used preceding flotation; the flotation tailings were, however, treated on tables. The results obtained were: Flotation recovery, 70.83 per cent.; table recovery, 7.51 per cent.; total recovery, 78.34 per cent.

Callow Flotation Installation

While the test mill was in operation, Mr. Callow, President of the General Engineering Co., advised the Inspiration company that he had invented and perfected a new flotation process which, in his opinion, would give as good, if not better, results than the machine of the Minerals Separation Co. As a consequence, arrangements were made to add a unit of machines of the Callow type. The flow sheet using Callow cells is illustrated in Fig. 4. It consisted of four rougher flotation cells, which served for the production of a low-grade concentrate, and another cell of the same construction which was supplied for the purpose of cleaning the concentrates made on the rougher cells. The tailings resulting from the cleaner cells were returned and mixed with the pulp entering the rougher cells. With the machine was furnished a Pachuca tank for the purpose of mixing flotation oil into the pulp entering the plant, should it be more desirable to do this in addition to or in place of feeding the oil to the grinding machines, as had proven useful both in the small and in the larger test plants. As a matter of fact, the use of the Pachuca tank was abandoned after several months' operation.

The Callow cells work on a different mechanical principle from that underlying the Minerals Separation Standard machine, inasmuch as no movable parts are used for the purpose of producing the fine dissemination of air with the pulp, which seems to be essential for any kind of flotation machine; while the Standard Minerals Separation machine depends on the operation of fast-revolving impellers, the dissemination is effected in the Callow machine by blowing air through the pores of a porous blanket, which forms the bottom of each cell.

The Callow machine, illustrated in another paper, is ingeniously simple. The feed enters at the upper end of the inclined bottom, while the tailings are discharged through the float valve at the lower end, the concentrates overflowing at the top of the flotation tanks. The porous bottom of the cell is set on an incline to make possible the treatment of feed containing coarse sand. The movement of such sand particles from the feed to the discharge end is thereby accelerated.

The mechanical principle of aeration and agitation by the admission of air through a porous medium, which has been termed "pneumatic flotation," was undoubtedly discovered by Mr. Callow[3] and his associates independently of other inventors, although an earlier patent had been taken out for the same thing in England by Minerals Separation (British Patent 10,929, May 3, 1910) and long before the installation of the Callow machine at this mill, Inspiration ore had been tested in New York by a pneumatic flotation machine constructed jointly by Messrs. Flinn and Towne.[4] The results of this test were so good that a small test plant of this system was installed at Inspiration.

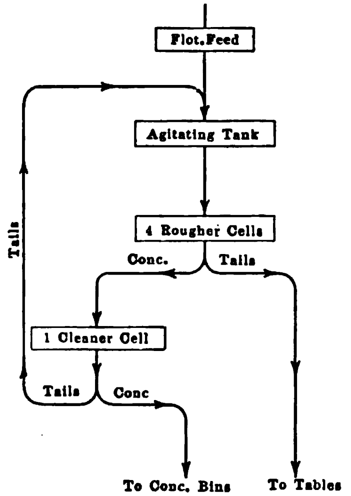

FIG. 4.—FLOW SHEET OF EXPERIMENTAL CALLOW FLOTATION PLANT.

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[3] Mr. Callow refers to his installation at the National Mill, Mullan, Idaho, April, 1914, as the first successful commercial plant utilizing the pneumatic principle.

[4] This test was made in the presence of Dr. L. D. Ricketts, Consulting Engineer, and W. D. Thornton, Vice-President of the Inspiration company. According to F. B. Flinn, a 600-ton plant of the Flinn-Towne flotation system was shipped to the Tezuitlan Copper Co. in Mexico in the spring of 1913. This could not be put into operation on account of the political conditions prevailing then. Otherwise, this would have been the first commercial pneumatic flotation plant. R. C. Canby informs me that some tests carried out by him suggested the idea to Messrs. Flinn and Towne of constructing a flotation machine to produce the desired fine dissemination of air by blowing air through a porous medium.

Flinn-Towne Installation

The Flinn-Towne machine, as mentioned, utilizes the pneumatic principle also, but the application is somewhat different. An illustration of a single Flinn-Towne machine is given in Fig. 5, while Fig. 6 shows the outline of the installation. The cells are constructed in the shape of cylindrical tanks, the bottoms of which are formed by the porous medium. The feed enters near the top in the center of the cylinder, while the tailings leave the machine through a center hole in the porous medium. In the Flinn-Towne installation, only one roughing machine was provided, while additional cells served for the purpose of cleaning

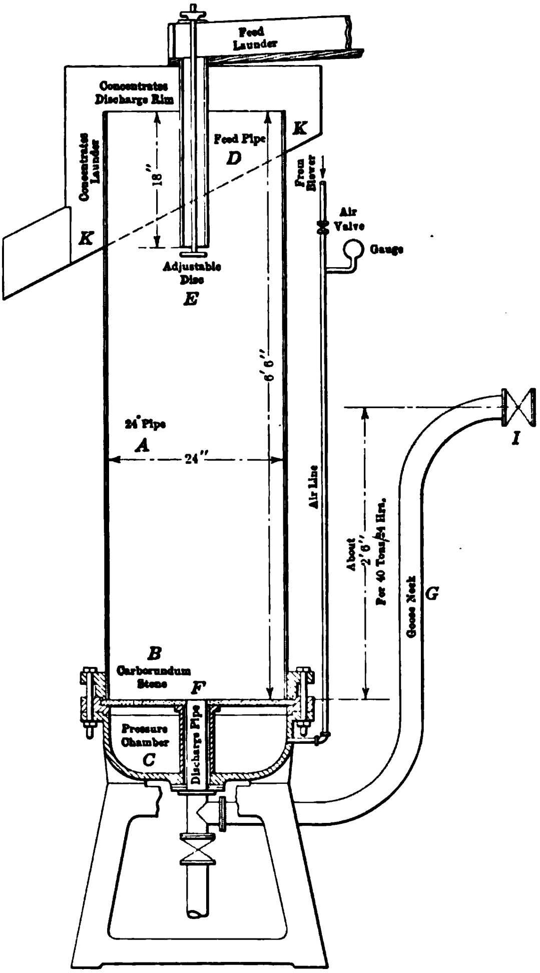

FIG. 5.—FLINN-TOWNE FLOTATION MACHINE.

both rougher tailings and rougher concentrates. The tailings produced by the concentrate cleaner cell were returned to the head of the roughing cell. In place of the canvas blanket of the Callow machine, Messrs. Flinn and Towne in the demonstration test at Inspiration used carborundum stones as the porous medium through which the air is injected into the pulp.

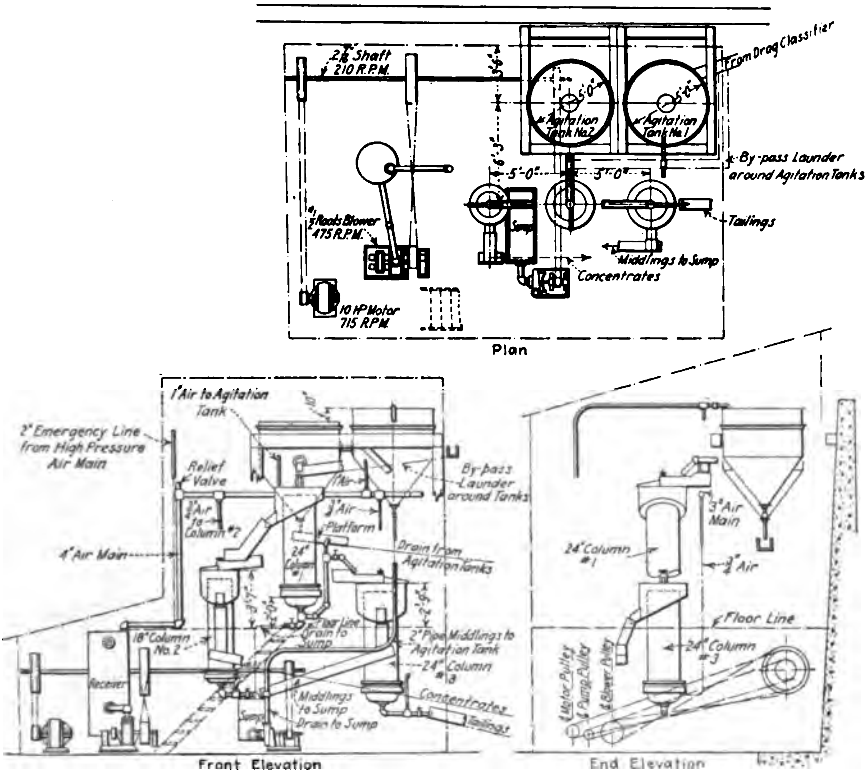

FIG. 6.—OUTLINE OF FLINN-TOWNE FLOTATION INSTALLATION.

Cole-Bergman Installation

The Flinn-Towne machine was withdrawn from the contest after a competitive test between this machine and others had run for several months, although the results obtained looked very encouraging. Its place was taken by a flotation machine designed by Messrs. Cole and Bergman.

Their machine is illustrated in Fig. 7. In principle the cells are similar to those of the Flinn-Towne machine. Evidently, the designers tried to improve on this system by mechanically developing the idea underlying the machine, and by changing one point, which in their opinion formed a weak part of the Flinn-Towne machine. This is the

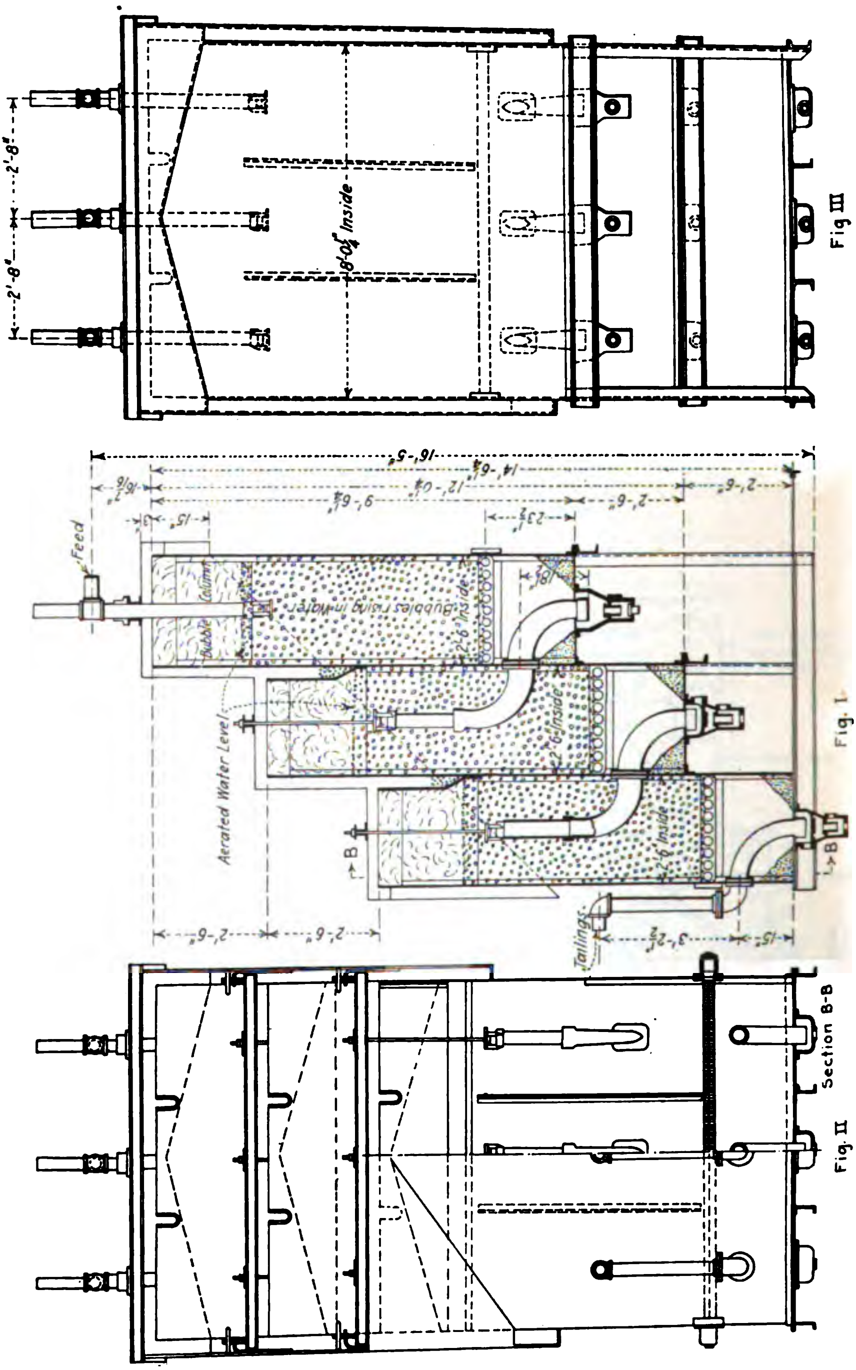

FIG. 7.—COLE-BERGMAN FROTHING CLASSIFIER.

construction of the porous diaphragm, which, as explained below, was formed by a round carborundum disk. While carrying on the test of the Flinn-Towne machine, it proved necessary occasionally to wash the carborundum stones by injecting a water pipe from the top and even by removing the stones and cleaning them with water, acids, etc. Messrs. Cole and Bergman ascribed this deficiency to the fact that on account of the horizontal position of the porous medium, sand had a tendency to lodge thereon and to impede or prevent the passage of air through the pores. They substituted, therefore, a system of perforated tubes covered with a suitable fabric, such as canvas or flannel. This led to the construction of a porous medium in the form of a grate, as shown in the illustrations. Their idea seems to be a very happy one, as it makes it possible to apply the flotation process to ore pulps containing comparatively coarse particles of sand; for instance, it seems possible to treat with this machine ore mixtures that contain sand particles as coarse as 10-mesh and perhaps even coarser. Using this machine, the millman is now in a position to treat slime by flotation without the necessity of removing it from admixture with sand. The Cole-Bergman machine has given good results in our tests, as have the rest of the machines utilizing the pneumatic principle. It was operated in conjunction with a smaller machine of the same type installed for the purpose of cleaning the concentrates produced on the larger machine.

Methods Followed in Competitive Tests

In the interest of a fair contest between the flotation machines of the different types, each machine was provided, as nearly as possible, with the same character of pulp as the rest. In the beginning this was accomplished by providing each machine with one or more pebble mills to which feed was sent in such quantities that a mill product of practically the same fineness resulted in each case. When the results of the competitive tests began to approach each other very closely, the equalization of the pulp furnished to the different machines was still improved upon by mixing the ground product discharged from all the mills, and sending it to a divider which permitted the division into as many parts as there were competing machines, and in any proportion desired. As it had been decided in former experiments not to install tables ahead of the flotation process, no preliminary table treatment was used during the competitive tests, but each flotation system was furnished with a set of tables for the retreatment of the flotation tailings. It developed that this was essential in trying to arrive at a fair valuation of the machines, as some of the machines produced tailings from which additional mineral could be extracted by the table treatment more easily than from the tailings of other flotation machines, the cause evidently being that flotation machines of one type have a tendency to save more of the finer particles, while flotation machines of another type permit of a better recovery of coarser grains. Those flotation machines that make a good recovery on the sand, but leave a larger percentage of mineral in the slime, will not benefit from a subsequent table treatment as much as those of the other kind.

Advantage of Pneumatic Flotation Machines

In the course of the tests at Inspiration, the fact became apparent that, by the application of the pneumatic principle, a higher recovery of the slime is made possible than without the utilization of injected air, although the Standard Minerals Separation machine also gives excellent results when the tonnage is reduced to a considerably lower figure than the machine is supposed to be able to treat economically.

Therefore, in cases where high power consumption is of little importance, the Standard Minerals Separation machine will fill the requirements of a flotation machine remarkably well. When, however, the power consumed has to be seriously considered, it seems advantageous in the light of the Inspiration experiments to make use of pneumatic flotation for the reason pointed out above, *i.e.*, that poor work of flotation machines on sands can be made up by a subsequent table treatment, while poor work on slime cannot. Within certain limits, it is more important to insist on machines effecting a good slime recovery than on machines making a good sand recovery.

Pneumatic Machines of Minerals Separation Co.

The Minerals Separation Co., realizing this, offered to put in their subaeration type of machine which added the advantages resulting from the use of injected air to the advantages which their Standard machines seemed to have in saving coarser mineral. This resulted in the construction and testing successively of two of their subaeration type machines, one in which the old spitzkastens still were retained, and another one in which they were dispensed with. As seen from the illustration, Fig. 8, provisions were made in the former machine to admit air to the pulp conduits carrying the pulp from the spitzkastens to the agitating compartments next in succession, in order to make this air available to lift mineral to the top of the spitzkastens where it can be skimmed off. The agitating compartments are closed at the top by covers set on an incline in such a way that the air rises to the top in the agitating compartments and is conducted to the spitzkasten, being thereby utilized for the purpose of carrying the mineral values.

This construction of the Minerals Separation Co. was, however, only an intermediate step toward the design of the second-named machine, a machine of greater simplicity, which is similar in principle to the one

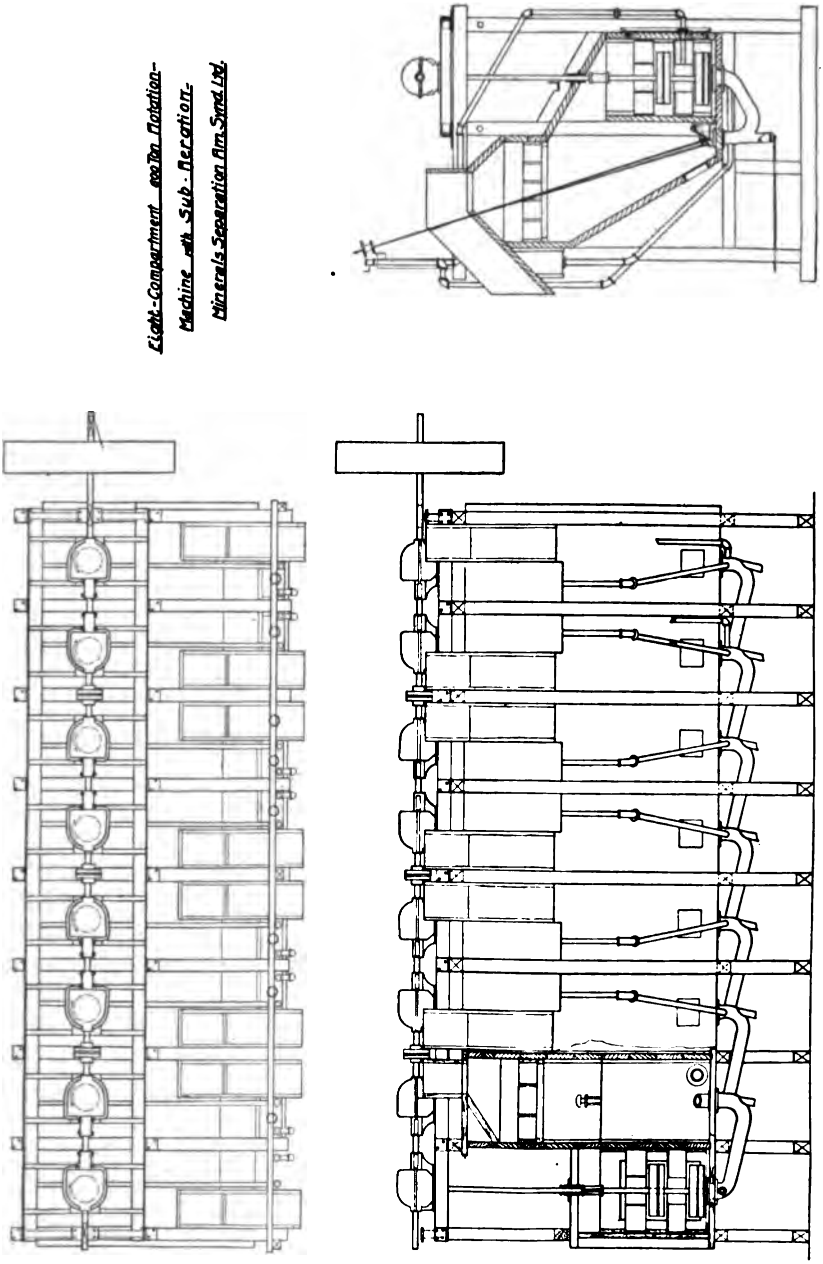
Eight-Compartment 600 Ton Flotation-
Machine with Sub-Aeration-
Minerals Separation Am. Synd. Ltd.

FIG. 8.—MINERALS SEPARATION SUB-AERATION TYPE OF FLOTATION MACHINE.

illustrated in Fig. 9. The machine consists of a long rectangular tank without any partitions, in which a number of agitators revolve, while air is

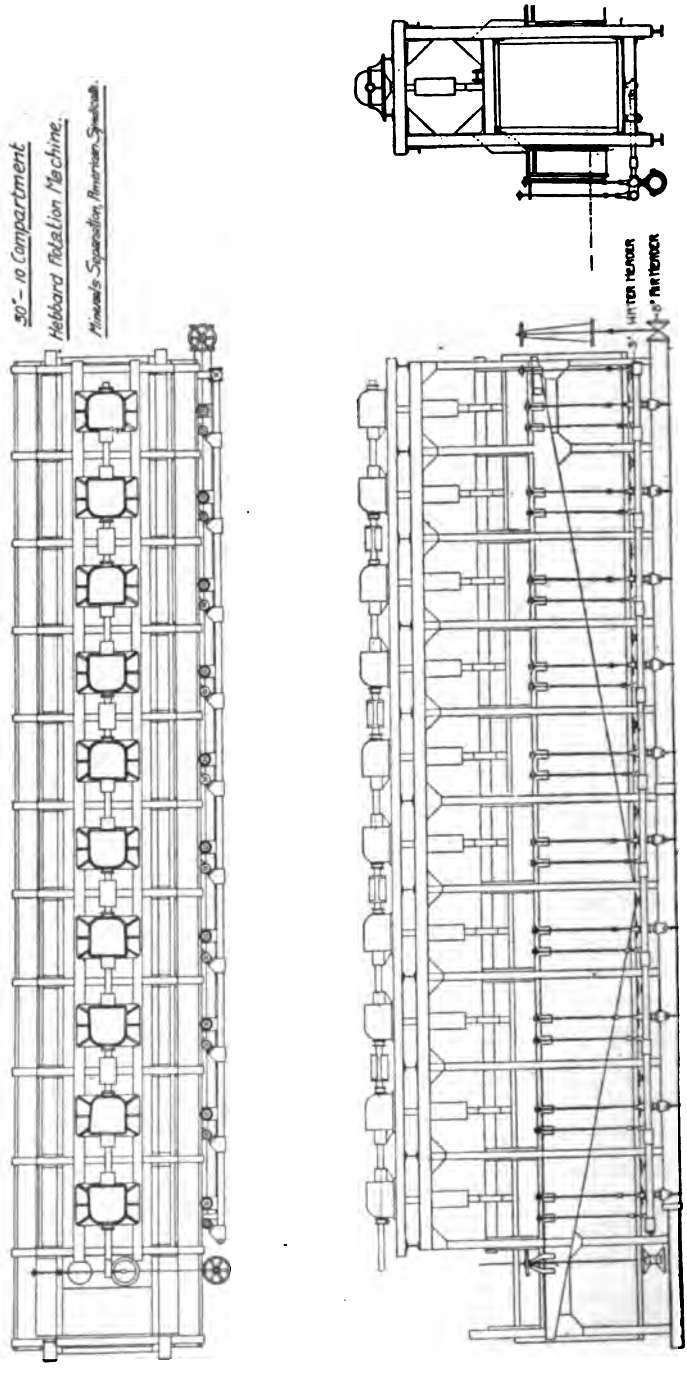
30'-10 Compartment
Hebbard Flotation Machine.
Minerals Separation, American Syndicate.
3" WATER HEADER
6" AIR HEADER

FIG. 9.—HEBBARD TYPE OF MINERALS SEPARATION MACHINE.

injected underneath each impeller. We call the machine the "Hebbard Type" Minerals Separation Machine.

In order to limit the agitation to the lower part of the machine and to provide an area of comparative quietness in the upper part, a system of baffles is arranged above the agitators. As a consequence, the froth is given a chance to separate out in the upper portion of the rectangular tank, and flows off on both sides lengthwise into launders provided for the purpose.

This machine marks an important progress in the development of the Minerals Separation flotation systems toward increased recovery of copper, at least as far as the Inspiration ores are concerned. In our tests it effected a higher recovery of the fine mineral particles than the old machine had accomplished. The machine also has the advantage of greater simplicity and relatively low power consumption. The only drawback that we have found in the operation of the machine, extending over a number of months, is that the agitating shafts, which are suspended from above, have a tendency to bend and, if not straightened, soon cause the impeller blades to strike against the baffles, breaking one or the other.[5]

Float Skimming Device

While the competitive tests of the different flotation machines were being conducted, it was observed that large quantities of mineral froth frequently appeared in the tailings launders. The idea suggested itself to remove the froth, thus increasing the recovery obtained in the flotation machines and concentrating tables by an additional small amount. After some experimenting, a way was found in which this could be easily accomplished. The tailings launder was widened and deepened for some distance. The tailings stream was forced to pass underneath a baffle on entering the widened space, and was also forced to travel under another baffle board at the end of the space, thus being made to rise through the restricted area formed by the tailings end baffle and the back-wall of the float-saving device, which is illustrated in Fig. 10.

Contrary to what might be expected, tailings sand of the fineness of Inspiration tailings will not lodge in the widened-out space unless the length is greater than a certain critical distance, nor will it choke the upward channel in the tailings end of the machine. When conditions demanded the construction of the float-saving device in greater length, it was found that the accumulation of sand, which has a tendency to form in the middle of the machine, can be avoided by arranging an additional baffle not reaching quite to the bottom in the center of the machine. If the machine is built longer yet, more than one intermediate baffle is necessary. The explanation for this behavior of the pulp, which at first glance might seem paradoxical, is that, on account of reduced passageway between the lower part of the baffles and the bottom of the machine, the pulp is forced to travel through at speed so high that it will not permit the settling out of sand. As a consequence, a hydrostatic head will establish itself between front and back of such a baffle and has to be taken care of in the design. It is found, however, in actual practice, that the hydrostatic head required to keep the sand from settling out by creating an accelerated current underneath the baffle is relatively small.

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[5] This trouble is alleviated in the Hebbard machines turned out more recently by the Minerals Separation Co. by the provision of suitable bearings for the impeller shafts in the bottoms of the machines.

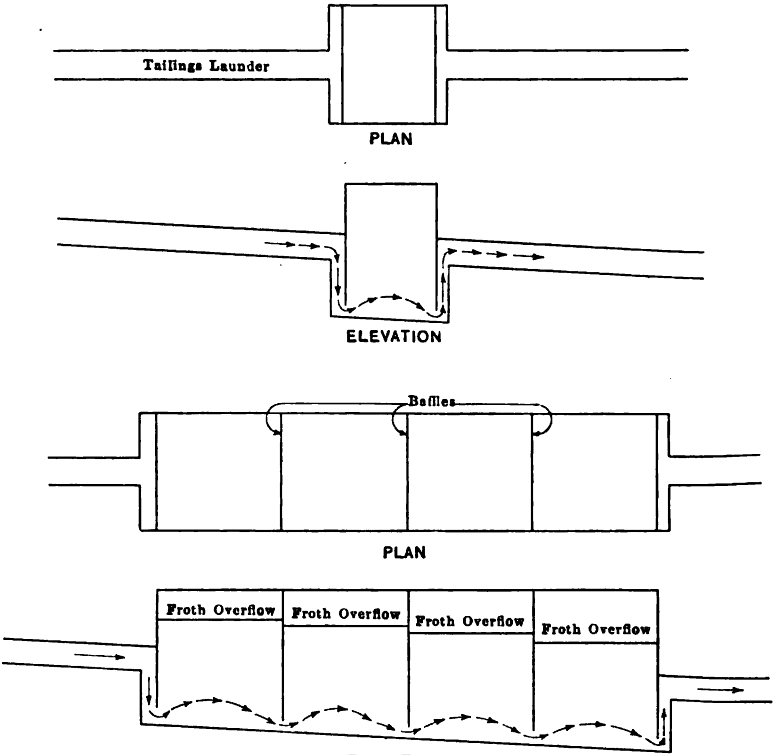

Fig. 10.—Diagram Showing Origin and Development of Inspiration Flotation Machine.

A similar consideration explains the fact that the upward tailings passage at the end of the machine has very little, if any, tendency to choke; to take care, however, of possible choke-ups that might be caused by the discharge of coarse rocks, pieces of wood, etc., into the pulp, it was found advisable to arrange some air jets in the bottom of the upward tailings passage to assist in clearing it whenever necessary.

Inspiration Flotation Machine

This device was in actual operation for several months at our test mill and made itself pay well by adding about 1 per cent. to the recovery actually obtained. The question then presented itself as to whether this device, which was successful in saving mineral escaping from the flotation

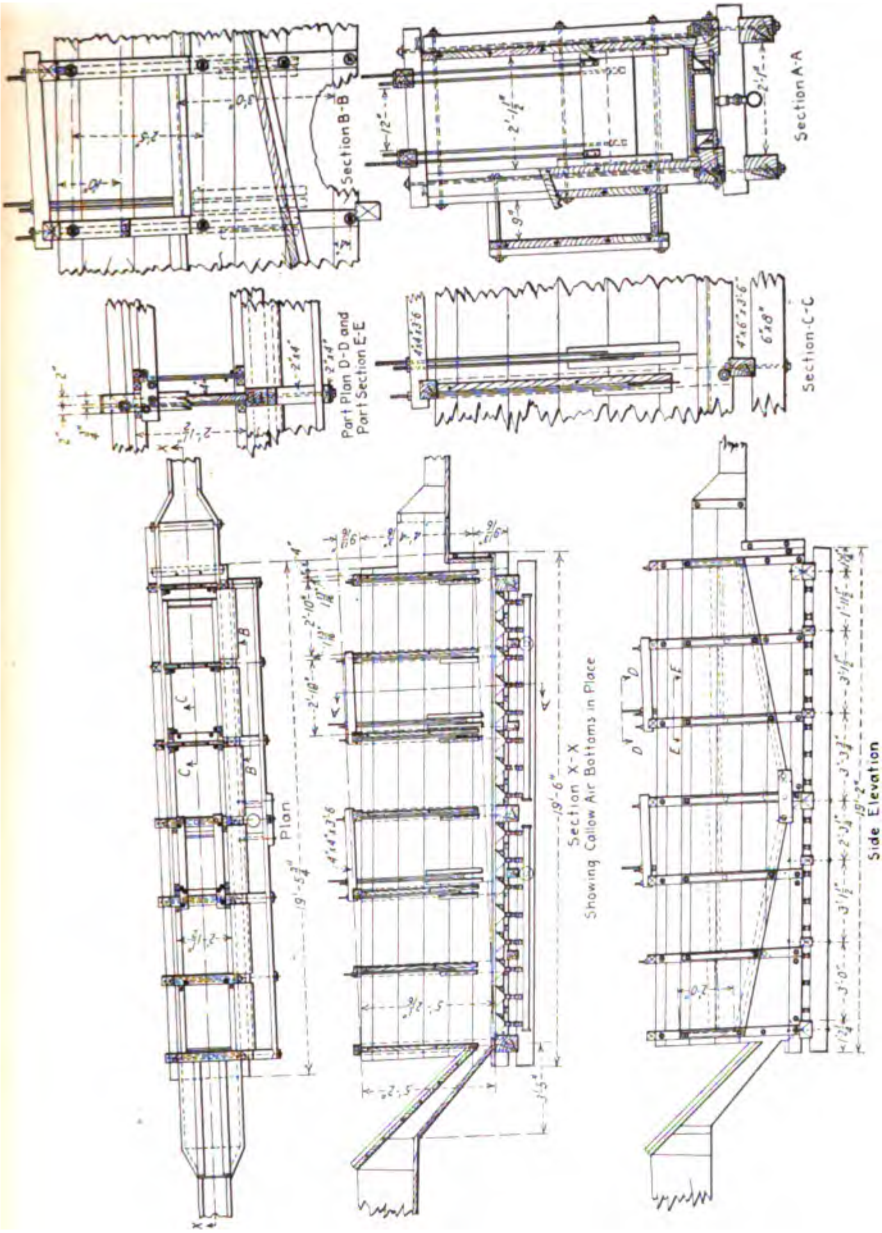

Fig. 11.—Experimental Inspiration Flotation Machine.

machines could not be transformed into a flotation machine itself by arranging for the injection of finely distributed air into the pulp passing through the machine. The development of the idea led to what we call our Experimental Inspiration Flotation Machine, which was built on a small scale, that is of a size to treat up to 100 tons in 24 hr., and was placed in competition with the other flotation machines already in existence. This machine is illustrated in Fig. 11. The air was injected through a porous bottom arranged in a way similar to the bottoms of the Callow and Flinn-Towne machines (as a matter of fact, a Callow bottom was used in the first tests) and the concentrate flowed over the edge on one side of the flotation tank. The mode of operation followed was also similar to the one developed for the other air machines. A low-grade concentrate was produced and retreated on a similar machine of the same construction, called the cleaner machine, the tailings from which were returned to the head of the roughing flotation machine. As will be seen from the drawing, the construction of this machine is extremely simple and comes pretty near to an ideal of Mr. Mills, our general manager, who prophesied that the flotation machine of the future would be nothing but a launder with provisions for injection of air. The method of effecting the concentration in two stages was also followed when the new types of the Minerals Separation machines were put into commission.

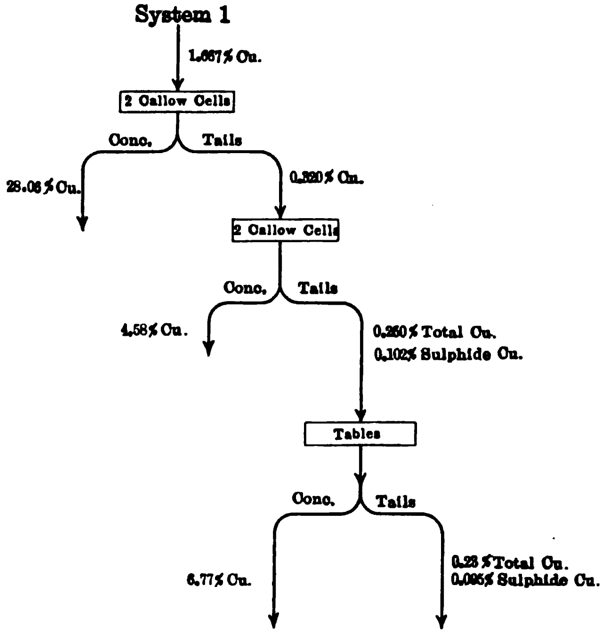

FIG. 12.—TESTS WITH CALLOW CELLS IN SERIES.

Arrangement of Callow Cells in Series

In principle, the Inspiration machine shows one difference from the Callow machine; viz., that instead of splitting the pulp between a great number of machines, it is forced to travel through a series of compartments in succession. This is the same policy that had also been followed in the construction of the Minerals Separation machines. The question then came up, whether it might not be of advantage to arrange the Callow cells also in series. To test out the idea, the flow sheet of the Callow plant was changed (see Fig. 12). The primary feed was sent to two of the cells, the other cells receiving the tailings from these primary cells.

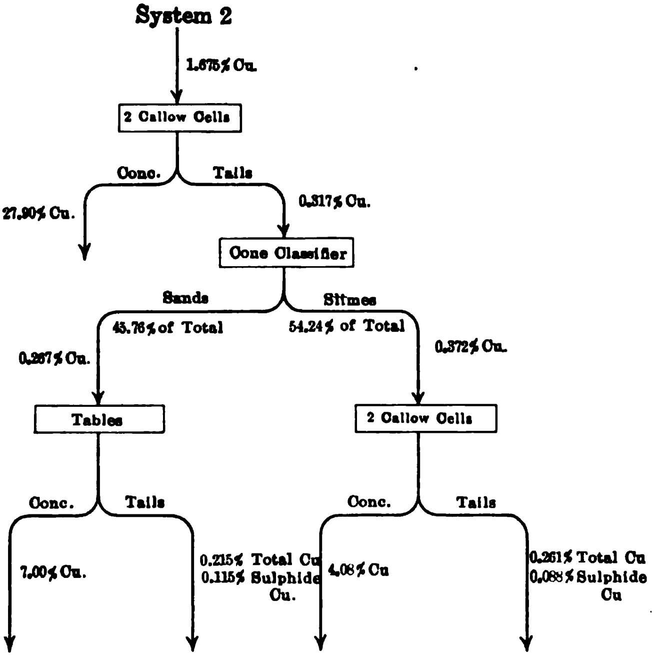

FIG. 13.—TESTS WITH CALLOW CELLS IN SERIES WITH CLASSIFIERS BETWEEN.

This arrangement showed at least no disadvantage over the multiple treatment; it seemed superior to the other system in the one respect that it showed at a glance whether the cells were operated well, because in that case most of the concentrate would be produced on the primary cells while the secondary cells would produce a rather light-colored froth and would add only a small fraction to the recovery made on the primary cells. Both primary and secondary concentrates were treated jointly in a cleaner cell, while the tailings from the secondary cells were conveyed to a number of concentrating tables for retreatment.

A variation of the latter flow sheet was also tested (compare Fig. 13).

Between the primary and secondary cells, a cone classifier was interposed for the purpose of making a separation between the sand and slime. The slime was sent to the secondary flotation machines, while the sand was passed to tables. This latter flow sheet was finally decided upon for the Callow installation in the large concentrator. Tests made with the object of establishing which of the two is superior, resulted in a tie. Some of the figures on this point are given in Table 2.

TABLE 2.—*Retreatment of Tailings from Two Callow Cells. Comparison of Results Obtained from Two Different Methods as Shown in Figs.* 12 *and* 13. *Screen Analysis of Feed and Tails of Secondary Callow Cells*

Mesh	System 1									System 2							
	Feed				Tails					Feed				Tails			
	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Sol. Cu	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Weight		Per Cent. Cu	Cu Contents
	Cum.	Indiv.			Cum.	Indiv.				Cum.	Indiv.			Cum.	Indiv.		
65	5.0	5.0	0.29	0.014	5.0	5.0	0.21	0.011	0.05								
100	19.0	14.0	0.25	0.035	19.2	14.2	0.15	0.021	0.06	3.8	3.8	0.20	0.007	4.0	4.0	0.17	0.007
150	32.2	13.2	0.29	0.039	32.4	13.2	0.16	0.021	0.10	10.6	6.8	0.25	0.017	10.8	6.8	0.14	0.010
200	42.4	10.2	0.33	0.034	42.8	10.4	0.21	0.022	0.11	18.6	8.0	0.30	0.024	19.0	8.2	0.25	0.021
	100.0	57.6	0.35	0.201	100.0	57.2	0.27	0.155	0.20	100.0	81.4	0.40	0.326	100.0	81.0	0.30	0.242
Assay calc.....			0.32	0.323	.....	....	0.23	0.230	0.15	.....	....	0.374	.....	.....	....	0.28	0.280
Assay direct....			0.32	.....	.....	....	0.23	.....	0.15	.....	....	0.370	.....	.....	....	0.26	.....

NOTE.—The concentrates from the flotation cells were in both cases retreated in half-size Callow cells, the retreatment tailings being returned to the head of the primary cells.

	Ton per Primary Cell
Tonnage rate in system 1............................	49.3
Tonnage rate in system 2............................	47.5

CONCLUSIONS.—No advantage is apparent in favor of either system. System to be installed should be decided from mechanical points.

Variations of Flow Sheet

In the way of other flow sheet variations, etc., only one thing was tried to any extent; that is, whether it is wise to send the cleaner tailings back to the head of the rougher machine for retreatment. In the limited time at our disposal, it proved impossible to substitute something better, but the chances are that the same holds true of flotation that is true for water concentration; viz., that middlings are snnt back for retreatment mainly because the designer does not know anything better to do with them. In principle it seems wrong to follow this practice, because, when heavy copper losses occur in concentrating machines which express themselves in high tailings, there is always a doubt as to whether this is due to poor recovery made in the primary treatment or to heavy losses incurred by middlings sent back for retreatment.

Coal Tar as Flotation Agent

It was to be expected, when the flotation process was installed in our test plant, that there would be ups and downs in the recovery because the process was rather new, especially in its application to chalcocite ores. For the month of June, 1914, the recovery obtained in the test mill showed a sudden drop, and the serious problem confronted us of establishing the cause and finding a remedy. Some of the flotation experts suggested that it might be due to the fact that a new shipment of cresylic acid which had been received and used at that time might not fill the specifications of being 98 per cent. pure. We did not feel competent to say whether the impurities actually amounted to more than 2 per cent. We were, however, inclined to think that perhaps cresylic acid, which is, as is well known, one of the products resulting from fractional distillation of coal tar, might not represent the fraction most suitable for the flotation of our ores. Having no coal tar available in the neighborhood, we proceeded to make some by distilling a sample of ordinary New Mexico soft coal and separating the tar thus formed into the fractions distilling off at different temperatures.

Our facilities for testing oils were limited at that time. The Minerals Separation Co.'s representative did not believe in small-scale tests, and for this reason did not recommend experiments with small testing machines. Nevertheless, it seemed desirable to have something with which to carry out small-scale laboratory experiments. Dr. Ricketts, who was aware of our troubles and realized the importance of such tests, was kind enough to send us a little electrically operated emulsifying machine, which served admirably for qualitative tests. We also built a testing machine based on the principle of the Standard Minerals Separation machine, with the difference, however, that instead of sending the pulp from one agitating compartment to a spitzkasten and then into another agitating compartment and spitzkasten, we made the pulp return from the first spitzkasten to the original agitator, forcing it to revolve in a closed circuit. Our laboratory machine is illustrated in Fig. 14. Lately, a machine based on the same principle has been put on the market and is sold by the Denver Fire-Clay Co. Thus, we had a chance to try the different fractions of our home-made coal tar.

The chemist who conducted these tests (Mueller) hit on the idea that it might be well, in addition to trying the different fractions, also to test the coal tar as a whole. The results were very surprising, since they showed that by the addition of crude coal tar we could effect a greater recovery than we were able to obtain by the use of highly refined cresylic acid. From this point dates our experience that it is better to use coal tar than soluble flotation agents like cresylic acid to save coarse mineral. Cresylic acid is an extraordinarily good agent for producing froth, but the froth which it produces does not seem to have as much carrying power for coarse mineral as that produced by coal tar. Not all coal tars are equally good for this purpose. Tests in laboratory machines easily show the difference between coal tars of different origin.

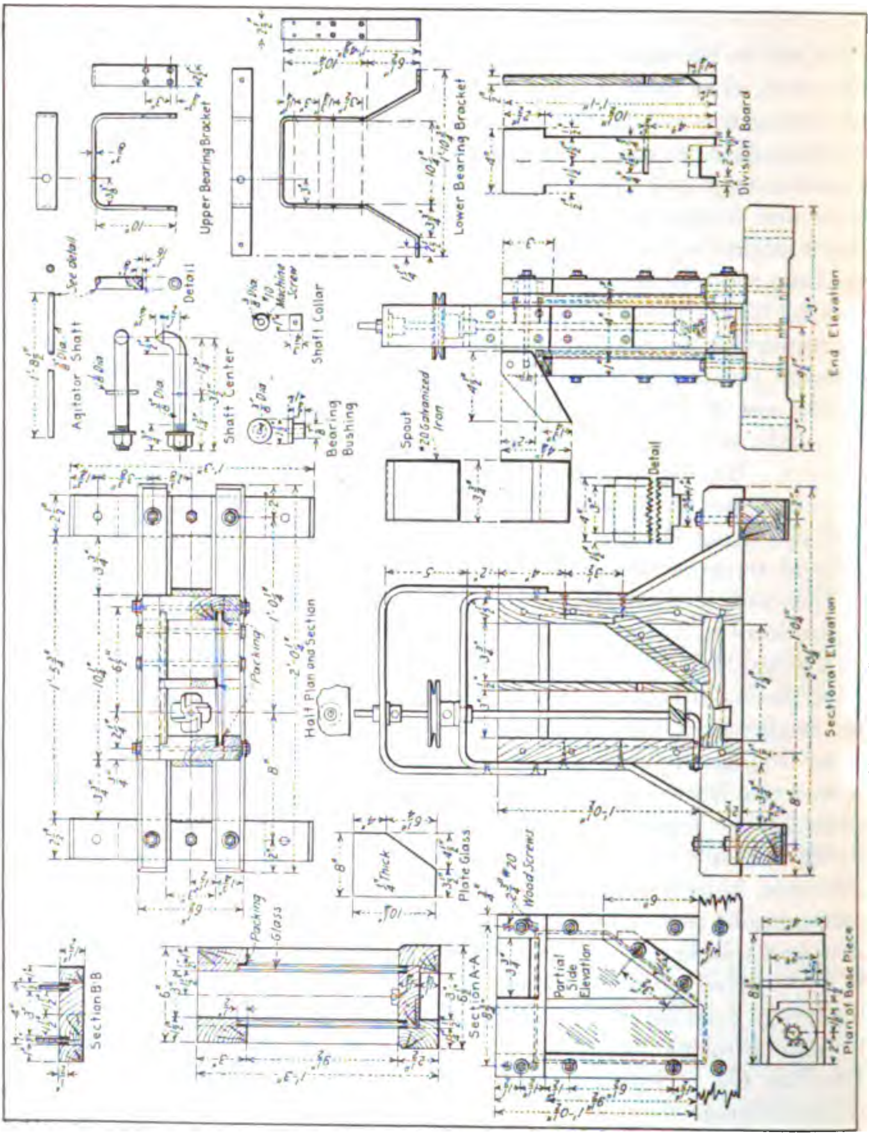

FIG. 14.—LABORATORY FLOTATION MACHINE.

The system that we used to carry out such tests has been described, although without authorization, by William A. Mueller, one of my former assistants who took part in these laboratory tests.[6]

It is difficult to utilize coal tar in plants using flotation supplementary to gravity concentration, on account of the fact that it is not easy to effect a good amalgamation of tar with the pulp in agitating tanks, and even in mechanical flotation machines. The use of coal tar lends itself very well indeed to the system of feeding tar into the grinding machines, a system that, as mentioned above, had been worked out in our small test mill and patented by Mr. Chapman.

The company is indebted to Mr. Callow for proving the merits of coal-tar creosote as a flotation agent by using it in his demonstration plant at Inspiration. After we had established the value of coal tar by laboratory tests, and while efforts were being made to obtain it commercially, he applied creosote successfully. We have continued to use it for a long time, mostly in combination with coal tar, and have only recently dropped it, as we find crude coal tar cheaper and better.

Experience with Primary Slime

After the first difficulty that we encountered in our large-scale tests had been solved by the introduction of coal tar into the flotation-oil mixture so far used, things went along fairly well for some time until a new difficulty was encountered. It happened that once in a while an abundant froth was produced on the flotation machines, but this froth seemed to have very little carrying power for the mineral contents of the ore and held mainly finely divided gangue. It was observed that this phenomenon occurred with special severity whenever the ores shipped to the test mill contained a large amount of fines and a small amount of coarse rock. It was attributed, therefore, to the presence of what may be called primary slime, *i.e.*, slime not formed by the crushing of ore in the mill, but originating from the mine. That the falling down of the flotation machines was caused by variation of the ores was proven by the fact that samples of the refractory ores when treated in the laboratory testing machine, gave as unsatisfactory results as the corresponding ore did in the big flotation machines. To demonstrate that the presence of the original slime caused the trouble, samples of the mill feed were separated by screening them on a 200-mesh screen. The oversize, when reduced to the proper fineness for flotation treatment, did not offer the least trouble and yielded a high-grade concentrate in the laboratory machines, while the undersize proved extremely refractory (refer to Table 3). The same experiment was repeated on a large scale, the products from a Marcy ball mill crushed to about 6-mesh being treated on a drag classifier. The oversize, after being reduced in pebble mills to the necessary fineness for flotation treatment, was sent to one group of flotation machines, while the overflow from the drag classifier was sent to another group. Fig. 15 illustrates the results of this experiment. While the reground oversize yielded remarkable lean tailings,

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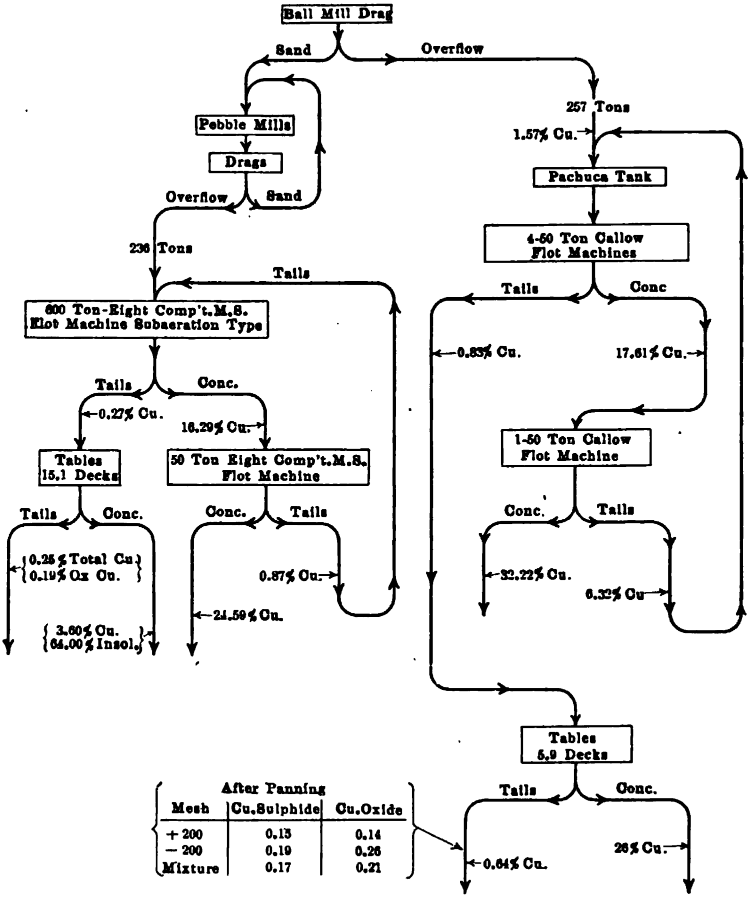
[6] William A. Mueller: Use of Coal Tar in Flotation, *Engineering and Mining Journal*, vol. 100, No. 15, p. 591 (Oct. 9, 1915).



Mesh	Cu.Sulphide	Cu.Oxide
+ 200	0.13	0.14
− 200	0.19	0.26
Mixture	0.17	0.21

FIG. 15.—TESTS TO DETERMINE EFFECT OF SEPARATE TREATMENT OF SLIME AND REGROUND SAND.

the product containing the primary slime could not be treated advantageously by flotation. We were almost on the point of concluding that in order to get the best results, a separation of the primary slime should be made in our large mill and flotation should not be entirely relied upon for the treatment of the slime. Before finally deciding on this point, however, some additional laboratory experiments were made to study the influence of the primary slime on flotation. These tests established

TABLE 3.—*Screen Analysis of Ore Drawn from Mill Bins and Flotation Tests of Different Screen Sizes*

Screen Size	Weights, Per Cent.		Per Cent. Total Copper	Copper Contents	Per Cent. Oxidised Copper	Contents Ox. Cu	Flotation Results		Recovery Per Cent.
	Cumul.	Indiv.					Tails	Midds. + Conc.	
Inches									
+ 1	30.8	30.8	1.05	0.324	0.13	0.40	0.22	12.8	80.5
+ ½	47.9	17.1	1.34	0.329	0.11	0.19	0.22	17.4	84.7
Mesh									
+ 4	70.9	23.0	1.44	0.332	0.12	0.28	0.17	17.1	89.0
+ 8	75.2	4.3	1.42	0.061	0.12	0.05	0.17	16.3	88.8
+ 14	80.3	5.1	1.77	0.068	0.13	0.07	0.26	16.8	86.7
+ 28	84.4	4.1	2.14	0.080	0.14	0.06	0.61	16.3	74.3
+ 48	87.5	3.1	2.62	0.081	0.19	0.06	0.72	11.3	77.5
+ 100	89.9	2.4	3.29	0.079	0.24	0.06	0.63	22.5	83.1
+ 200	91.9	2.0	3.16	0.063	0.27	0.05	0.60	20.1	83.5
− 200	100.0	8.1	1.53	0.124	0.43	0.35	1.57	1.48	None
	.....	100.0	1.44	1.441	0.16	0.157			

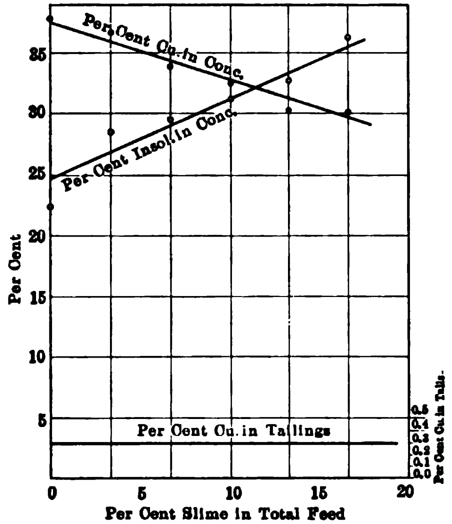


FIG. 16.—INFLUENCE ON FLOTATION OF MIXING PRIMARY SLIME AND REGROUND SAND.

The + 1-in. and + ½-in sizes resulting from the screen analysis were ground to 10 mesh, mixed with varying amounts of − 200 material and the mixtures ground with the usual amounts of oil in the laboratory ball mills. After this preliminary treatment, the samples were treated in the laboratory flotation machines.

one fact quite well, which is, that when refractory slime is mixed with a sufficient quantity of coarse ore ground to the necessary fineness, in other words, when the percentage of primary slime in the flotation feed is kept low, the slime loses its refractory character. I do not mean to say that by reducing the quantity of primary slime, the deleterious influence is reduced in proportion so that it cannot be detected as easily, but conclude from our tests that when the flotation feed contains a sufficient percentage of comminuted coarse rock, the primary slime contained in it can be treated more advantageously than it could by itself. Some tests bearing on this point are illustrated in Fig. 16. They indicate that as high an amount as 20 per cent. primary slime may be mixed with ground coarse ore without causing increased tailing losses.

The concentrates, according to this set of tests, seem to have a tendency, however, to carry more insoluble matter with an increasing percentage of slime.

Influence of Iron on Flotation

While these tests were in progress, we made another accidental discovery which proved very helpful to us. In our tests on the most economical way of reducing the ore to the fineness necessary for flotation we had, among other machines, a ball mill in competition with pebble mills. In the ball mill, steel balls performed the duty that in pebble mills was done by flint pebbles.

For a while, the ball-mill discharge was treated on one flotation machine, while the pebble-mill discharge was treated on a group of others. While this flow sheet was being followed, we thought we noted that a flotation machine treating the ball-mill product showed the influence of the primary slime to a lesser extent than the flotation machine treating the pebble-mill product. In a discussion with Dr. Ricketts and Mr. Mills, the question was raised as to whether the iron introduced in the mill pulp by the attrition of the balls might not have something to do with the fact. The question was accordingly made the subject of some laboratory experiments. The results of a series of such experiments are represented in Table 4, and proved conclusively that the iron had a beneficial influence on flotation in counteracting the harmful effect of the primary slime. This discovery was one of the inducements for installing ball mills in the big concentrator, while originally pebble mills had been considered for this purpose.

We have not yet reached a point where we can safely give the reason for the action of the iron introduced into the flotation pulp. It is sure, from the experiments referred to, that the same results, as by grinding with balls, could be obtained by introducing the iron in finely divided form, say in the form of filings, into a pebble-mill pulp. We supposed for a while, and I am not yet certain that this supposition is incorrect, that the metallic iron might react on the impurities contained in solution in the mill water and introduced therein with the primary slime. We find, as a matter of fact, that our ores contain very little in the nature of

TABLE 4.—*Effects of Iron and Other Solids on the Flotation of Refractory Copper Ores*

Test No.	Grams Ore	Per Cent. Copper	Grams Copper	Concentrates			Recovery Per Cent.	Remarks
				Grams	Per Cent. Cu	Grams Cu		
F21	750	2.01	15.07	45	23.66	10.65	70.7	Added 10 g. iron filings.
F22	750	2.01	15.07	47	29.52	11.52	76.4	Added 10 g. iron filings.
F27	750	2.01	15.07	43	27.10	11.65	77.5	Added 2 g. iron filings.
F28	750	2.01	15.07	47	26.90	12.64	84.0	Added 2 g. iron filings.
F45	750	2.01	15.07	51	23.84	12.16	80.7	Added 10 g. iron filings.
F46	750	2.01	15.07	48	25.60	12.29	81.5	Added 10 g. iron filings.
F47	750	2.01	15.07	27	8.80	2.38	15.8	Blank with no solids added.
F48	750	2.01	15.07	28	8.34	2.34	15.5	Blank with no solids added.
F49	750	2.01	15.07	62	20.14	12.49	83.0	Added 10 g. miscellaneous iron filings from shops.
F50	750	2.01	15.07	63	20.10	12.66	84.0	Added 10 g. miscellaneous iron filings from shops.
F51	750	2.01	15.07	60	18.82	11.29	75.0	Same as F49 and F50 by different observer.
F52	750	2.01	15.07	62	19.54	12.11	80.5	Same as F49 and F50 by different observer.
F53	750	2.01	15.07	29	6.16	1.79	11.9	Blank with no solids added.
F54	750	2.01	15.07	30	7.96	2.39	15.9	Blank with no solids added.
F55	750	2.01	15.07	59	21.08	12.44	82.7	Added 10 g. iron filings.
F56	750	2.01	15.07	56	25.80	14.45	96.0	Added 10 g. iron filings.
F64	750	2.01	15.07	65	19.52	12.69	81.9	Ground in mill with steel balls instead of pebbles.
L27	750	1.10	12.75	37	28.42	10.51	81.7	Blank on good flotation ore.
L28	750	1.10	12.75	34	28.46	9.85	78.3	Blank on good flotation ore.
L29	750	1.10	12.75	76	5.66	4.30	33.4	Identical conditions as L27 and L28 but added 10 g. zinc filings.
L30	750	1.10	12.75	83	5.06	4.20	32.9	Identical conditions as L27 and L28 but added 10 g. zinc filings.

soluble salts, and that whatever they do contain is mainly confined to the primary slime. For this reason, in laboratory tests we have tried repeatedly to substitute pure water for the water contained in the mill pulp. In every case, we have noted some improvements in the results obtained. We have also found that when we separate the water from the refractory pulp, treat it with iron filings, and add it to the original pulp again, we get a certain improvement in the recovery, but we have not been able to get an improvement equally as good as that obtained by direct introduction of finely divided iron into the pulp. For this reason, we have often thought that the effect of iron is physical rather than chemical in character. The iron exists in the pulp, at least partly, in the metallic form, as can be proven by the use of the magnet. If necessary, the effect of the iron could be increased by removing the iron contained in the tailings pulp by means of electromagnets and returning it to the mills or the flotation machines. Although we have considered the possibility of applying this fact commercially, we have not yet done so.

I have been told by several people that they have tested the influence of finely divided iron on their ores and have not obtained any improvement whatever. This shows, evidently, that metallic iron is not a universal remedy for all flotation troubles, but as far as our primary slime is concerned, our experience leaves no doubt about its usefulness, and I believe the figures quoted above are positive enough to bear out my statement. As a curiosity I have included in Table 4 some figures showing the influence of the addition of zinc filings to ore that without the addition does not offer any difficulty in the flotation treatment.

Influence of Chemicals on Primary Slime

We also found that the harmful influence of primary slime can be counteracted by the introduction of certain chemicals; for instance, we have used advantageously sodium hydroxide or potassium cyanide. The use of sodium hydroxide was recommended to us by the Minerals Separation Co., which, I understand, applied it first in the plant of the Caucasus Copper Co.

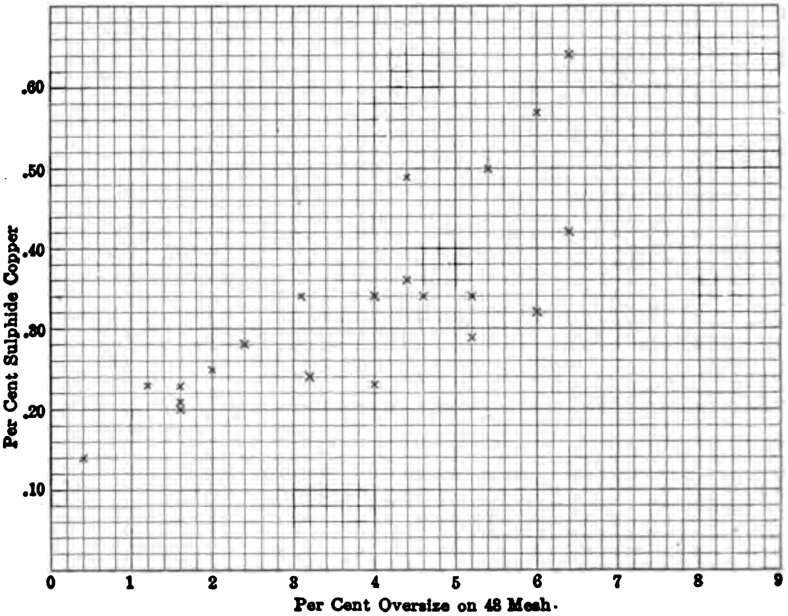


FIG. 17.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW FLOTATION TAILING.

Fineness of Crushing Desired

Tests to determine the most advantageous fineness for treatment of the ore according to our flow sheet, *i.e.*, flotation machines followed by tables, naturally formed an essential part of our tests. Results of this character are plotted in Figs. 17 and 18 and seem to indicate that to get the best results grinding should be carried to such a point that not more than about 1 or 2 per cent. of the pulp will remain on a 48-mesh screen (Tyler type). The grinding at our large concentrator is at present carried closely to this point.

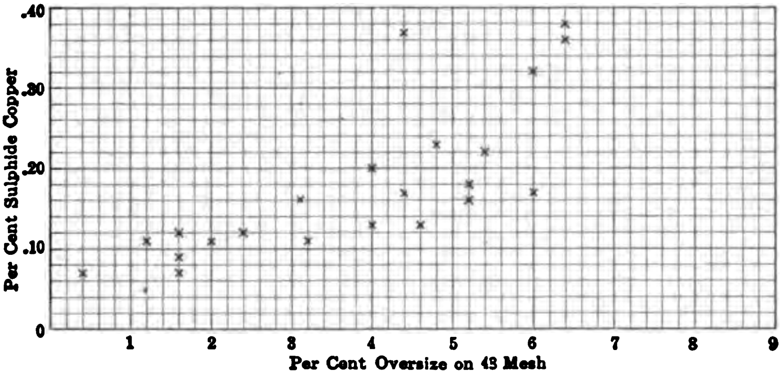


FIG. 18.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW TABLE TAILING.

OPERATION OF LARGE CONCENTRATOR

Flow Sheet of Callow Sections

The first four sections of the Commercial Concentrator of the Inspiration company were equipped with Callow flotation machines. The flow sheet followed in these sections is illustrated in Fig. 19. As will be seen from this illustration, the ore, after having been reduced to the desired fineness, and having been oiled in the Marcy ball mills, is sent to 12 (in other sections 8) Callow cells. The tailings from these primary cells are sent to a drag classifier designed by Mr. Burch, which effects a separation of sand and slime. The slime is retreated in 12 (in other sections 16) additional Callow cells, while the sand is sent to a hydraulic classifier and then to tables. The concentrates made both on the primary and the secondary Callow cells are retreated in a group of four Callow cleaning cells from which the tailings are returned to the head of the primary Callow cells. On the tables of the Deister Machine Co. a separation is made between concentrates and tailings, but no middlings are produced and no part of them is reground. This, however, is probably only a temporary arrangement, the idea being that a suitable regrinding and reconcentration process will be installed later, after having been worked out by extensive tests.

Flow Sheet of Inspiration Sections

Thirteen sections of the Inspiration concentrator are equipped with Inspiration flotation machines. The flow sheet of these sections is represented in Fig. 20. The ore, after having been crushed and oiled, passes to the head of the flotation machines and traverses their whole length. The resulting tailings are split, by the same kind of a drag as mentioned before, into a sand and a slime product. The slime product is run to waste. This is thought permissible because, after having passed through the whole length of the flotation machine, the slime has been impoverished to such an extent that retreatment seems unnecessary.

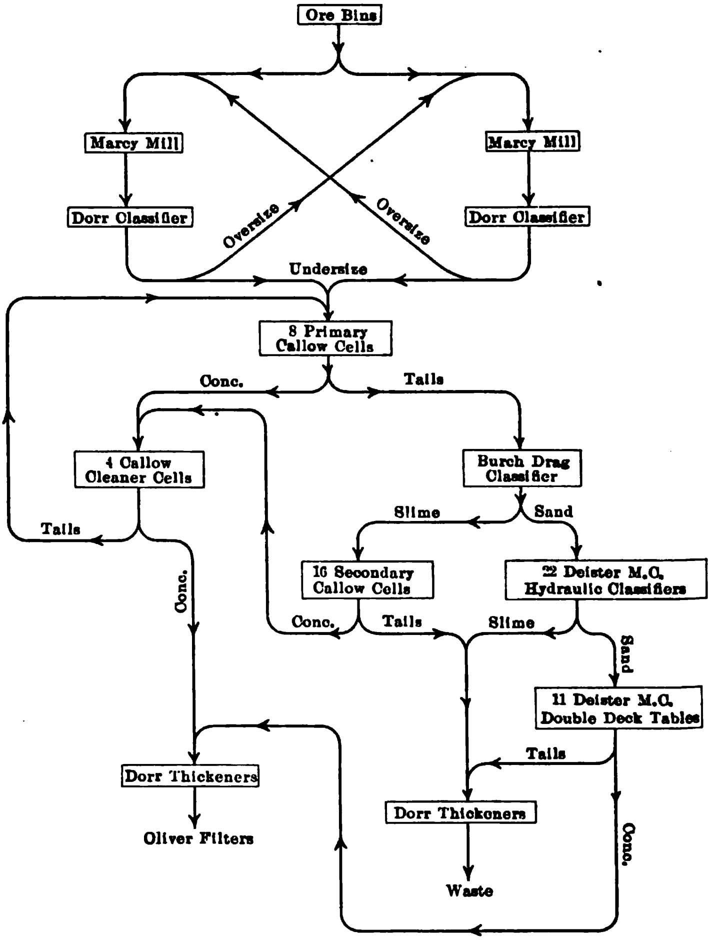


FIG. 19.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH CALLOW FLOTATION MACHINES.

The sand product undergoes the same retreatment as that described for Callow sections.

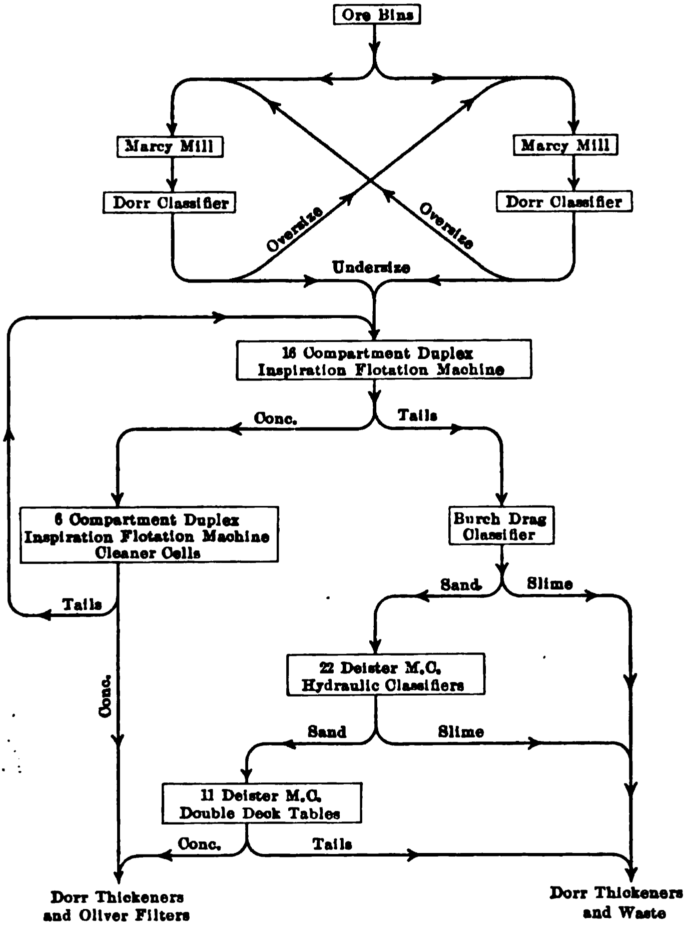

Ore Bins
Marcy Mill
Marcy Mill
Dorr Classifier
Dorr Classifier
Oversize
Oversize
Undersize
16 Compartment Duplex Inspiration Flotation Machine
Conc.
Tails
6 Compartment Duplex Inspiration Flotation Machine Cleaner Cells
Burch Drag Classifier
Tails
Sand
Slime
Conc.
22 Deister M.C. Hydraulic Classifiers
Sand
Slime
11 Deister M.C. Double Deck Tables
Conc.
Tails
Dorr Thickeners and Oliver Filters
Dorr Thickeners and Waste

FIG. 20.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH INSPIRATION FLOTATION MACHINES.

Inspiration Flotation Machine of Steel Construction

Fig. 21 shows the drawing of an Inspiration flotation machine of steel construction as designed by Mr. Burch's office. At the end of each section of eight compartments, a pulp overflow is provided which regulates the level of the pulp in the preceding compartments of the flotation machine. The subdivision into two sections of eight is made in the interest of closer regulation of the pulp levels. It had been feared that if no such subdivisions were made, and the regulation of the levels accomplished at the tail end only, appreciable fluctuations might take place in the compartments near the feed end due to changes in the volume of pulp treated, and might prove highly undesirable.

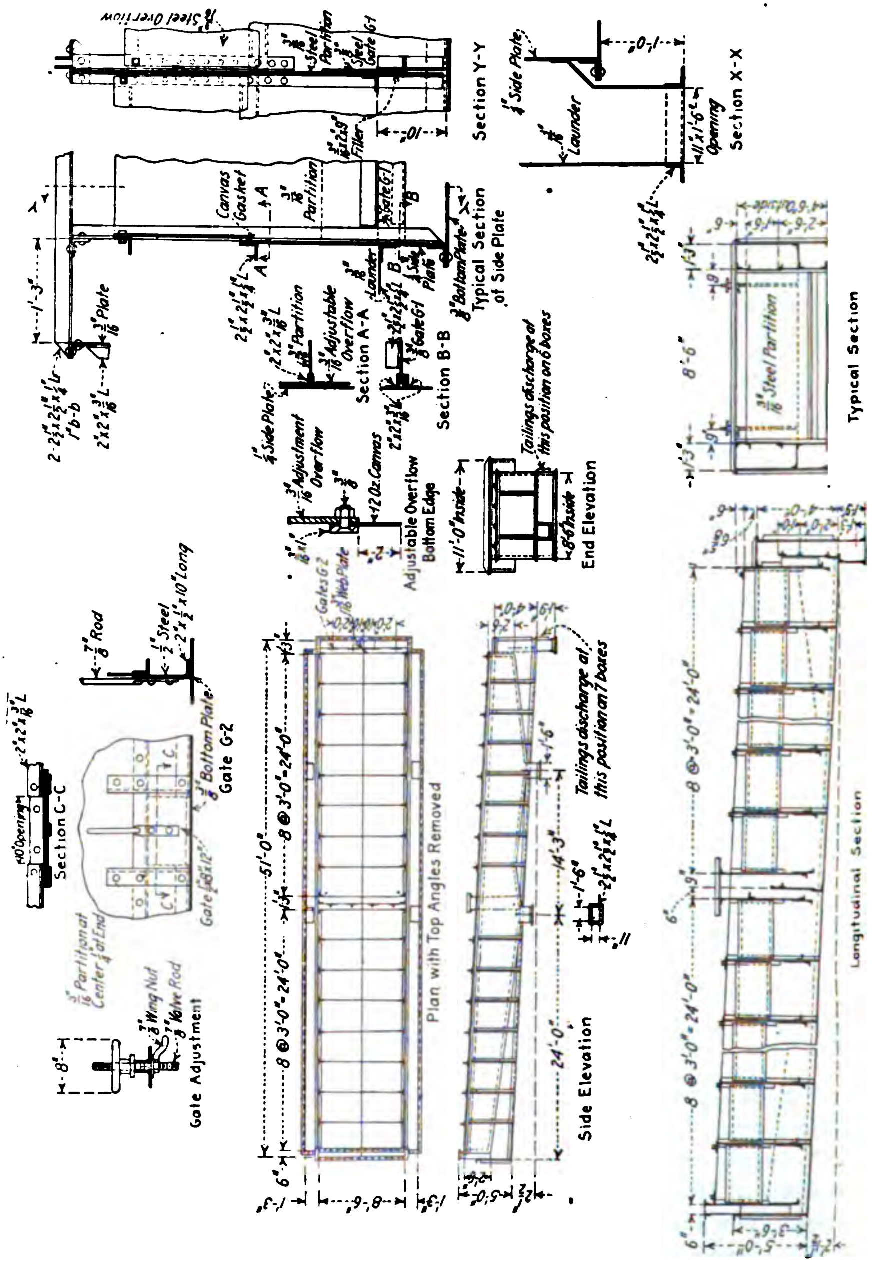


FIG. 21.—INSPIRATION ROUGHER FLOTATION CELLS OF STEEL CONSTRUCTION.

One point in the construction of the Inspiration flotation machines in which we take considerable pride, is the method in which the air is introduced. The machines have solid bottoms; all pipe connections are made from above and are at all times visible. The porous bottom rests on

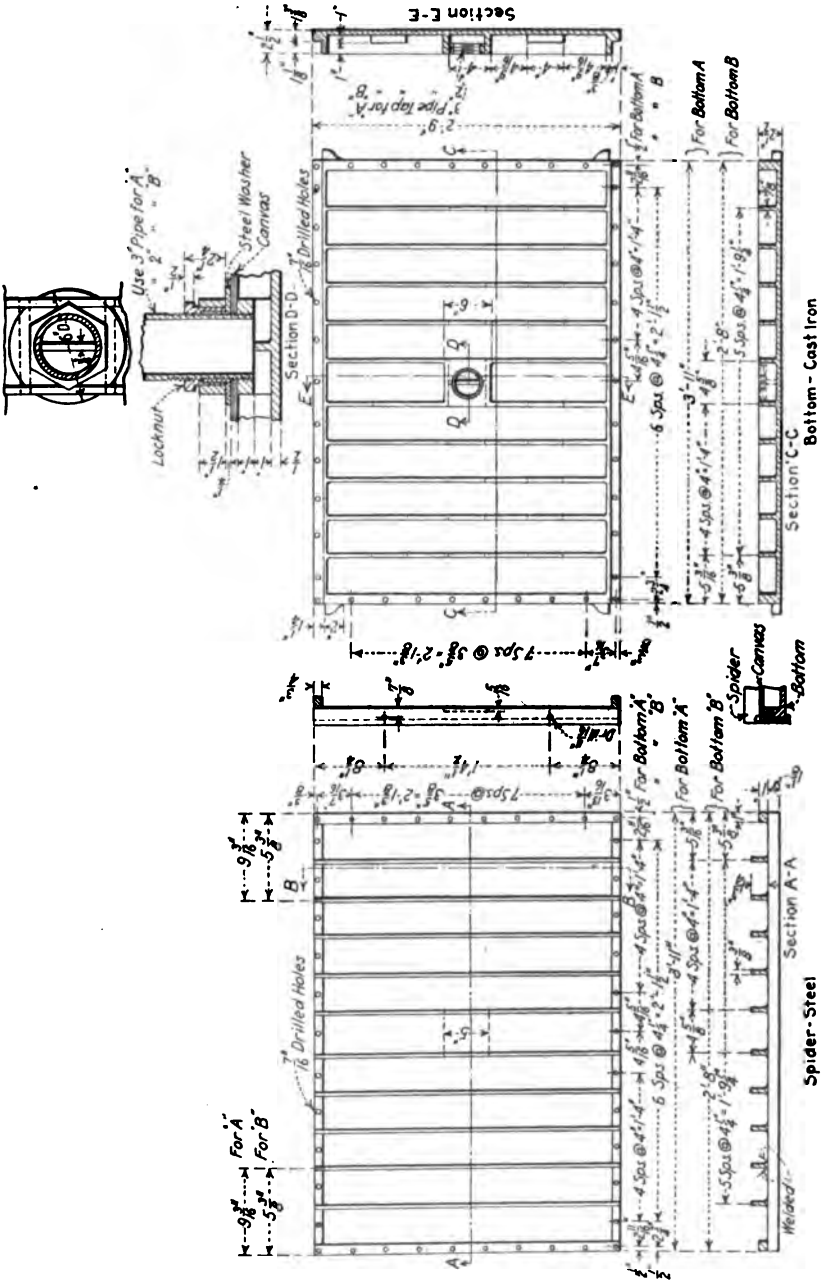


FIG. 22.—INSPIRATION FLOTATION MACHINE BOTTOMS.

top of the regular bottom of the machine and is introduced from above. It is illustrated in Fig. 22. As will be seen, it consists of a lower air chamber having partitions running lengthwise, which, however, do not extend down to the bottom of the air chamber and, for this reason, do not interfere with the interchange of air between the narrow compartments thus formed. A grate, which forms the upper part of the porous bottom, can be fastened to the lower part by a number of bolts. Before putting them together, a porous medium (for instance, canvas) is placed between the two pieces. The air enters through a pipe from the top which screws into the lower casting. Arrangements are made, of course, to secure an air-tight joint where the pipe passes through the porous medium. When assembled, the bars of the upper grate form channels in the lengthwise direction of the machine. The bars are the only parts that protrude above the canvas and, on account of their arrangement longitudinally, they do not interfere with the passage of pulp through the machine.

The cleaner cells consist of six compartments in series. Both rougher and cleaner cells are divided by a partition in the middle, running the whole length of the machine. The operator is, therefore, enabled to throw all the feed on one side of the machine when desired, thus permitting repairs to be made on the compartments of the other side whenever necessary.

In the beginning, it had been supposed that throwing all the feed on one side of the flotation machine might seriously overload that side, and for this reason, in the first sections installed, provisions were made to permit removing and replacing the porous bottom from any one compartment without interrupting the flow of pulp. As far as lifting out the bottom is concerned, this, of course, can be done in any case, but while one of the bottoms is withdrawn sand will pile up to a certain extent in the middle of the compartment. To remove this sand before returning the porous bottom, a system of perforated water pipes was installed under the air chamber. They allow the washing out of the bottom of the machine by the application of water pressure.

Minerals Separation Section

One section of the concentrator is equipped with machines of the Minerals Separation Co., Hebbard type. The flow sheet, as illustrated in Fig. 23, is practically the same as the one shown in the Inspiration sections. The only difference is that two cleaner cells in parallel are used in place of the one cleaner cell of the Inspiration type. The drawings of the machine are reproduced in Fig. 9.

Filter Plant

It is necessary to reduce the ore to a certain fineness before the mineral contents of the feed can be treated by flotation, and as fine concentrates have the characteristic of retaining water with greater tenacity than coarse concentrates, a filter plant is a necessary adjunct of a flotation plant. In the Inspiration mill, both the flotation and table concentrates are sent to Dorr thickening tanks, five of them being 60 ft. in diameter and three of them 80 ft., representing a total area of about 29,217 sq. ft.,

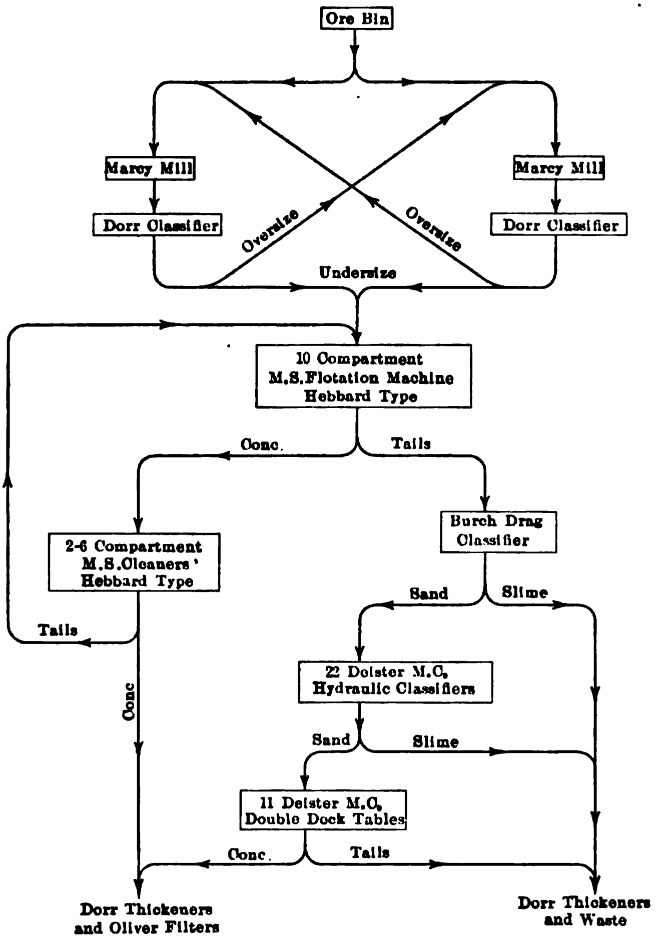


FIG. 23.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTION EQUIPPED WITH MINERALS SEPARATION FLOTATION MACHINES.

or, as the daily production of concentrates amounts to about 600 tons, a settling area of around 48.7 sq. ft. per ton of concentrate sent to the settling tanks. The settling tanks are each provided with a double ring of high baffle boards to prevent the foam that forms on the top from contaminating the overflow. The settling used to be carried out in two steps, some of the tanks serving as preliminary settling tanks, while others resettled the overflow from the preliminary tanks. At present we operate all the concentrate settling tanks in parallel. The spigot discharges feed six Oliver filters. The ratio of solids to water is about 1.65 to 1 at this point. The filters reduce the concentrate moisture to an average of about 17 per cent. of the wet pulp. This figure varies, however, within rather wide limits from day to day. The moisture contents of the concentrate seem to depend in the first place on the percentage of insoluble material contained in the concentrates, as will be seen from the graphic representation of the relation between the two things in Fig. 24. It will be noted that a rise in the percentage of insoluble matter generally corresponds to a rise in the concentrate moisture and *vice versa*. A moisture of 17 per cent. is, of course, in excess of what seems desirable in view of the fact that the smelter penalizes water contained in the concentrates. For this reason, several attempts have been made to reduce the moisture. The first plan followed was to inject steam into the filter tank, with the object of heating the pulp. An appreciable increase in the capacity resulted from this, while no marked reduction of the moisture contents took place. The same effect was obtained by the addition of slacked lime, which we tried lately in our Oliver filters. It shows itself very readily in the formation of a thicker filter cake, thus causing a larger output of the filters. We have, however, not been able to effect a reduction in moisture by the use of lime. We have adopted the addition of lime to the filter pulp in regular operation on account of its decided benefit as far as the capacity is concerned. As a rule, five filters can handle the whole output of the concentrator; that is, each 12 by 12-ft. Oliver filter has a capacity of about 120 tons in 24 hr.

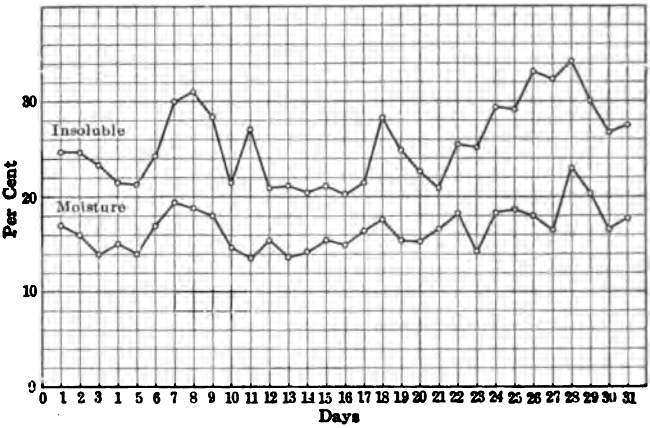


FIG. 24.—CURVES SHOWING RELATION BETWEEN MOISTURE CONTENTS AND INSOLUBLE MATTER IN CONCENTRATES. OCTOBER, 1915.

RESULTS OBTAINED IN COMMERCIAL CONCENTRATOR

On account of the fact that the concentrator operated by the Inspiration company has only just left the stage of construction, it is not possible to give results obtained in the operation of it as a whole. As, however, an interval of several months has elapsed between the starting of the first and the 18th section, we are in a position to give at least preliminary results derived in actual operation on a fairly large scale.

Tonnage

Each of the sections of the Inspiration concentrator has a rated capacity of 800 tons; at present most of the sections exceed 900 tons actual capacity. Ordinarily, this tonnage is treated on one Standard duplex Inspiration machine consisting of 16 double compartments, while the concentrates are cleaned on a cleaner cell with six double compartments. The size of the roughing compartments is 3 ft. by 4 ft. 3 in.; the size of the cleaner compartments is 3 ft. by 3 ft. Accordingly, the total area of combined rougher and cleaner cells is 492 sq. ft. Assuming the tonnage handled to be 800 tons, this makes a tonnage treated of 1.62 tons per day per square foot of surface. In the course of construction, the erection of flotation machines did not always keep pace with the erection of the grinding mills. For this and other reasons, it has frequently been necessary to combine the product from the two grinding sections in one flotation section; in other words, to throw an overload of 100 per cent. on the flotation machines. Our experience is, that overloading the machines by 100 per cent., or raising the tonnage to 3.24 tons per square foot of surface, does not increase the tailing losses very materially.

It follows, therefore, that the machines handle an overload without serious additional copper loss, and it has been our experience that there is no mechanical trouble whatever, connected with overloading the flotation machines to such an extent. We have also had to resort to overloading the Callow cells, and have likewise found that the recovery does not fall off seriously.

For the Minerals Separation machine of the Hebbard type, which is installed in our Section No. 10, we have no figures of a similar character.

Air and Power Consumption

We have found that three blowers, of a capacity of 8,000 cu. ft. per minute each, are sufficient to supply air for six sections of flotation machines. Accordingly, one section requires 4,000 cu. ft. of air per minute. As the flotation machines in an Inspiration Section have a surface of 492 ft., it follows that the air consumption per square foot of porous surface is 11.8 cu. ft. per minute; the maximum pressure required is 4½ lb. at the blowers and practically the same at the flotation machines, as there is no serious loss of pressure in the air-conduits. The power consumed for furnishing air has been determined carefully by wattmeter readings of the motors furnishing power for the blowers. The actual power consumed is 87.5 kw. per section; or, figuring again with the rated section capacity of 800 tons, 2.63 kw.-hr. per ton of ore treated. We have not been able to make separate tests for the Callow and Inspiration sections, but assume that the air consumption is approximately the same in both. We have no figures yet for the Hebbard type machine. To get the total amount of power consumed in connection with the flotation machines, a small amount of power, probably about 10 kw. or 0.30 kw.-hr. per ton of ore, has to be added to the power consumed for the production of air on account of the fact that the cleaner tailings are pumped back to the rougher cells and retreated, and that in the Callow sections the slime overflow of the drag classifiers is repumped to the secondary flotation cells. In the Inspiration sections, 0.425 sq. ft. of actual open porous surface and 0.615 sq. ft. of flotation machine area are provided for the treatment of 1 ton of ore in 24 hr.

Floor Space

The total floor space of the flotation floors amounts to 25,200 sq. ft., or 1.75 sq. ft. per ton of ore treated in 24 hr. (rated capacity). The total floor space occupied by the concentrator proper (not including, however, the crushing plant located at the mine) is 81,900 sq. ft. or 5.68 sq. ft. per ton of ore, and the total floor space, if the settling-tank installation outside the main mill building is also included, is 250,890 sq. ft., or 17.4 sq. ft. per ton of ore (rated capacity).

Water Consumption

The water added to the pulp where it reaches the flotation machines is about 3 tons of water for each ton of solid pulp. The subsequent treatment, in our case, requires a further addition of nearly 3 tons of water per ton of dry ore. However, as it is carried out on pulp that has been deprived of its slime contents by passing it over drag classifiers, the water thus entering the table tailings can be easily removed again. This is accomplished in our case by settling the table tailings separately from the slime tailings. The settling of the sand tailings is effected in a settling tank of home construction, consisting of a rectangular box with a sloping bottom into which a number of Caldecott cones are inserted. The resulting overflow is not quite clear, but is made so by resettling in three 60-ft. Dorr tanks.

Out of the total quantity of water actually added to the ore during its passage through the mill, amounting approximately to 6 tons of water per ton of ore treated, as mentioned above, nearly 3 tons, *i.e.*, the amount added during the table treatment, can, as explained above, be reclaimed rather easily. For the reclamation of the rest of the water, settling ponds, of the well-known construction in which the reservoir is formed by building a retaining wall of sand across a gulch, are resorted to in addition to Dorr thickening tanks.

Recovery

The recovery obtainable by the application of our milling process is determined entirely by the composition of the ore, *i.e.*, by the relation between sulphide and oxide copper contained in the ore. Our average sulphide copper extraction has been 90.39 per cent. for the months of March, April and May, 1916, the last months for which figures were available at the writing of this paper. A certain recovery of the oxide-copper minerals, especially carbonates, is made in the flotation process as well as in the gravity concentration process. The percentage of such mineral recovery is low, probably about 25 per cent. For this reason, the recovery obtainable on ore containing a high amount of oxide, such as surface ore, is correspondingly lower. We have worked in the laboratory with the object in view of increasing the oxide recovery; for instance, by adding certain chemicals to the flotation pulp. An account of the results obtained will be found below, but we have not yet applied this method to an operating scale, nor have we decided on using one of the other methods applicable for this purpose, such as leaching.

Table 5 gives average screen analyses of the feed and the general tailings of the Inspiration concentrator for the months of March, April and May. A segregation is made in the copper assay between sulphide and oxide copper, because, considering the present stage of the art, we feel satisfied with our mill work whenever the sulphide copper content of the mill tailings is low. As will be seen from the tabulations, a better recovery is made on the −200 material than on the coarser constituents of the ore, which proves the point that for ores of the character of Inspiration ore sliming is not to be feared since the introduction of the flotation process.

Oil Consumption

Experience has shown that in the flotation treatment of our ores, we consume flotation agents up to 1½ lb. per ton of ore. At present, the oil mixture contains about 95 per cent. crude coal tar and a little less than 5 per cent. of oils derived from the dry distillation of wood.

The different tars that we have tested during the operation of our mill have shown greatly varying qualities as far as their flotation value is concerned. The first tar that we tested and made use of was homemade from domestic coal, and happened to be a very serviceable flotation agent. Since that time, we have tested tars from several States and have found some that are suitable for our purposes, while others are less so, and still others even entirely unsatisfactory. We have obtained satisfactory tar products from New Mexico, Colorado, Missouri and Illinois. These States furnish at present as much as we need for our consumption. For awhile, it seemed possible that we might have to import from a long distance the large quantities of tar that we require. During that period we tried to find substitutes, and looked especially toward the utilization of fuel oil for this purpose, but we have not been able to get as good results with any kind of fuel oil as with crude coal tar.

TABLE 5.—*Average Screen Analyses for the Months of March, April and May,* 1916, *of Flotation Feed and General Tails*

	Flotation Feed				General Tails							
Mesh	Per Cent. Weight		Copper Contents		Per Cent. Weight		Sulphide Copper Contents		Oxide Copper Contents		Total Copper Contents	
	Cum.	Indiv.	Per Cent.	Grams	Cum.	Indiv.	Per Cent.	Grams	Per Cent.	Grams	Per Cent.	Grams
+ 65	9.5	9.5	0.45	0.042	9.5	9.5	0.18	0.017	0.12	0.011	0.30	0.028
+100	21.2	11.7	0.86	0.101	21.2	11.7	0.19	0.023	0.14	0.016	0.33	0.039
+150	33.5	12.3	1.91	0.235	33.5	12.8	0.11	0.014	0.19	0.023	0.30	0.037
+200	39.2	5.7	2.69	0.154	39.2	5.7	0.14	0.008	0.19	0.011	0.34	0.019
−200	....	60.8	1.85	1.125	....	60.8	0.06	0.036	0.47	0.286	0.53	0.322
Totals......	....	100.0	....	1.657	....	100.0	....	0.098	....	0.347	....	0.445
Assay direct	....	....	1.62	....	....	....	0.101	....	0.318	....	0.419	
Oxide......	....	....	0.36			.						

Our experience is, that we can get along with coal tar alone. It is beneficial, however, to add wood-distillation products in small quantities, for instance, those containing pine oil. While coal tar makes a very strong and heavy froth, such as appears to be required to keep coarse mineral particles in suspension, the wood-distillation products have the characteristic of producing a multitude of froth bubbles, such as seem necessary to furnish the large surface required to save the very fine mineral particles. Because the finer ore particles (slime), on account of their small diameter, expose a large surface, it is evidently necessary to produce a correspondingly large surface of froth in order to save them by flotation.

Operating Cost for Flotation

The number of men necessary for the operation of large flotation machines is remarkably small. At the Inspiration plant, one operator supervises four sections of flotation machines. Two Mexican helpers assist him in washing the bottoms, thus insuring a free passage of air through the porous medium. At the prevailing high prices of American and Mexican labor, this means an expense of somewhat more than 1.5 c. per ton of ore treated. The total expenses representing flotation proper were as follows for the months of March, April and May, 1916:

	Cents per Ton
Labor	1.62
Flotation oils	1.65
Other supplies	0.35
Power	2.14
Total	5.76

The subsequent table treatment of flotation tailings, the filter treatment of the concentrates and other operations connected with the process of concentration, belong more or less to flotation treatment, and their expense should also be considered when the cost of the flotation process is to be established. The total milling cost, exclusive of crushing and grinding, has been for the past few months in the neighborhood of 20 c. When the cost of crushing and grinding is included, the cost is about 40 c. per ton of ore. Royalties for the use of the flotation process are not included in any of these cost figures.

Discussion of Results Obtained

The criticism has occasionally been raised against the Inspiration company that they were slow in deciding on the design of a concentrator. The reason for such slowness was, of course, that the development of the flotation process coincided with the period during which the fundamental points in the design of the concentrator had to be settled. The management, therefore, thought it best to be very conservative in installing standard concentrator equipment, which, in case the new process held what it seemed to promise, might prove to be entirely superfluous. The main point of interest, therefore, is whether it was wise for the company to wait instead of installing a gravity concentrating plant as had been originally considered.

As far as the comparison of a flotation plant, as at present operated by the company, with the gravity-concentration plant, as originally considered, is concerned, the recoveries actually obtained in the company's flotation plant are so much higher than those indicated by tests in the original test mill as being obtainable in a gravity-concentration plant on ore of the same character, that there cannot be the least doubt which system is the better.

In discussing the second question which comes up in this connection, whether, for ores of the character of Inspiration ore, gravity concentration supplemented by flotation would be preferable to the simple flotation process that has been adopted by the Inspiration company, the crucial point is this: Can the slime be reduced to lower copper contents by a combined process than by the process followed here? The reduction of the copper in the sand is, as millmen know, simply a question of fine grinding combined with suitable concentration.

In support of the principle followed by the Inspiration company, I desire to call attention to the screen analyses of our general tailings represented in Table 5. They show that the sulphide copper left in the −200-mesh material is very low.

In addition, the floor space required for a plant like this is materially smaller than for a plant of the combination plan, which also means that the construction cost is much lower; and, further, the operating costs are low by reason of the greater simplicity of design. I hope these facts are sufficient to decide the point at issue as far as the ore of this district is concerned.

I hope that our neighbors of the Miami company, whose mill was built before the flotation process was known, will uphold me in my statements. If this point is admitted, it will endorse the principle followed in the design of the Inspiration mill, of doing away with reduction in stages and concentration in stages.

It must be granted that the settling of flotation-concentrate pulp requires extensive floor space, as shown by the figures referred to above, and that settling and subsequent filtering absorb a certain fraction of the mill operating costs. However, as a ratio of concentration in the Inspiration mill is 25 into 1, only 1/25 of the ore is thus treated. The actual expenses increase, of course, proportionately for ore which does not permit concentration with such a high ratio as the ore existing in the Inspiration mine. Therefore, there will be a point beyond which a different system of concentration is preferable. Just what the critical ratio of concentration is, must be determined by calculation and testing based on individual conditions.

PROSPECTS FOR FUTURE DEVELOPMENT OF THE FLOTATION PROCESS

The flotation process is in its infancy. For this reason, therefore, our concentrator must be necessarily in the first stages of its development. In what direction future changes may take place, is perhaps indicated by tests which have been made partly on a laboratory scale and partly on a somewhat larger scale, but which have not yet been incorporated into our regular milling process. Of these latent developments, I will try to give an outline in the following:

Porous-Bottom Experiments

The porous bottom is, as one may imagine, the most essential part of a pneumatic-flotation machine. Our experience with the porous bottoms of the different constructions brought out very clearly the principal difficulty attached to them, which is, that the pores have a tendency to contract gradually and thereby to retard the passage of air through them. This tendency was more pronounced in the solid porous bottoms employed in the Flinn-Towne flotation machine than it was, for instance, in those of the Callow type, although the latter also show a tendency in this direction. Our first supposition was that the choking was due to the fact that the air entering below the blankets carried particles of dust, which would settle in the fine pores and reduce their area. Indeed, a canvas blanket will, after a certain length of service as a porous medium, always show some ring-shaped spots of dark color opposite the air inlets, clearly indicating that a deposition of dust particles on the blanket actually does take place. To make sure of this point we cut out round disks from a Callow blanket that had been used for some time and investigated their porosity by using them as porous bottoms in a glass tube standing in a vertical position. Air under pressure could be applied to an air chamber located underneath these disks, and the air passing through the porous blanket could be measured by a gas meter. The quantity of air discharged through the porous medium offers a measure of the porosity of the blanket, and for this reason, the velocity or speed with which the counter of the gas meter revolves, gives an indication of the porosity of the porous disks being tested. To our surprise, we found that the darkest points of the blanket were not those of lowest porosity. On the contrary, the points farthest away from the air inlet showed the greatest tendency to choke. An explanation of this paradoxical behavior seems to be offered by the fact that an air blanket is kept in a state of more or less agitation near the air inlet (in the Callow machine this happens to be a point remote from the places where it is held rigid) while farthest away from this point the blanket assumes a state of comparative rest. Incrustations, due perhaps to the presence of soluble salts in the water in conjunction with fine slime, always form to a greater or less extent in the top layer of the blanket. Evidently, the agitation counteracts the formation of the incrustation, while there is no such counteracting influence in the portions which are essentially at rest. For this reason, we concluded that a solid porous material is not suitable as a diaphragm in a flotation machine of the pneumatic type, if a bottom of long life is required. As a matter of fact, the experience of everybody who experimented with solid bottoms seems to have pointed in the same direction. Mr. Cole for a while tested out carborundum tubes in his machine. We tried carborundum stones in the flotation machine of the Inspiration type and abandoned them, and I believe that even Messrs. Flinn and Towne have, in the meantime, given up the solid bottom of their original design.

A necessary condition for a serviceable flotation bottom appears, therefore, that the porous medium be of a flexible nature. The 4-ply canvas stitched every half inch or so which Mr. Callow's first cells contained and which we have used for considerable time in the Inspiration machines, seems to answer this purpose fairly well. We find, however, that to keep it in good working condition and prevent incrustations from forming on the top, we have to clean it frequently. This is done by dipping an iron pipe connected with a water hose into the compartments and sweeping the canvas bottom with the jet of water discharging from the lower end of the pipe. The canvas blankets seem to last for about 6 months at the most. As they are inexpensive, the replacing of a bottom after that time is not a serious item in the operating costs. The giving out of the canvas is due to the wear caused by the frequent cleaning. The top layer wears out first, the holes created by the stitching forming nuclei for the formation of larger holes. By the time the top layer has a number of holes the canvas blanket is generally discarded. In the interest of greater economy, we intend giving up interstitching the layers of canvas. We are trying to decide whether it is better to use single sheets of thicker fabric or to use canvas similar to the kind that we have been using and to put several layers on top of one another without interstitching them. The latter has the advantage of requiring the discarding of only one layer, when it becomes defective.

There will always be some tendency to form incrustations so long as canvas is used for flotation mediums. Their formation will be entirely prevented only by substituting an altogether different material. We have made experiments in this direction. One of my former assistants, R. H. Haskell, deserves credit for suggesting them. For instance, we substituted for the canvas blankets, thin rubber sheets perforated with a multitude of needle holes and obtained an excellent froth. The objection to their use is that their life is limited. When sheets of rubber of an increased thickness are used, the needle holes require too much pressure to form openings of sufficient size for the passage of air, and to make a thick rubber sheet suitable for this purpose, slits several millimeters long have to be substituted for needle holes. We have had one or two rubber bottoms of this design in operation, but, just at present we are not ready to substitute them for canvas blankets. We also tried a blanket made from a material that goes under the name of sponge rubber and can be produced with rather fine texture. We have not been able, however, to obtain lastingly good results from the use of this medium. Furthermore, we tried a woven fabric containing rubber threads in one direction and threads of cotton or the like in the other direction and a rubberized canvas made by the Goodrich Rubber Co. We are not prepared to use any of these materials on a commercial scale.

The advantage of rubber should be, in the first place, that on account of its smoothness it would have less tendency than canvas to permit the formation of incrustations. Besides, an elastic medium should have the additional advantage of avoiding the danger of catching small sand or slime particles in the pores of the medium, as an expansion of the medium (that may be effected, for instance, by increasing the pressure) would widen the pores and remove such particles. We think that our experimental work in this direction is encouraging.

Raising the Grade of Concentrates

The recovery that it is possible to effect in a flotation plant depends largely on the grade of concentrate desired. With a low grade of concentrate, low tailings can be made, but when a high grade of concentrate is stipulated, increased tailing losses cannot be avoided. A question that suggests itself in this connection, and which we have tried to answer by laboratory experiments is, "How can we raise the grade of our concentrates—that is, reduce the percentage of insoluble matter contained in them—without entailing additional copper losses?" We know from laboratory experiments that this can be done by expensive methods—for instance, by heating the solutions—but such a procedure would be undesirable from an economical standpoint. Experience has shown us that concentrate produced in the first compartments of the cleaner cells is always freer from insoluble matter than the concentrate produced in the last compartments. The problem then resolves itself into finding a suitable cleaning process for the concentrate from the last compartments of the cleaning cells. By treating this low-grade concentrate hot, with the addition of caustic soda, we have been able to separate it into a high-grade concentrate and fairly low tailings. This method necessitates only the expense of heating a small fraction of the pulp and may be a commercial possibility.

Recovering Carbonates by Flotation

Another subject on which we have spent considerable time in our laboratory is the problem of recovering copper carbonates by flotation. When we started our flotation plant, we discovered, to our astonishment, that the machines not only saved a high percentage of copper sulphide but that they also recovered some of the carbonates. Ever since that time, we have tried to find means of improving the carbonate recovery.

In the first place, we studied all of the oils that seemed to have a tendency to cause the flotation of such minerals. Later on, we tried other means in addition to the variation of the oils. One way in which copper carbonates and similar minerals might be recovered was outlined by Alfred Schwartz in his United States Patent No. 807,501. The process consists in first artificially producing a sulphide coating on such oxidized minerals by the introduction into the pulp of soluble sulphides, and then adding suitable "oils" and effecting the flotation. If it were possible to thus chemically produce coatings of sulphide identical with the surface of the minerals formed by nature, this process would work well, as evidently the nature of the surface is the only characteristic that determines whether a mineral will float or not.

The Minerals Separation Co. owns a number of patents covering this subject. Their English Patent No. 26,019, issued to Sulman and Picard, describes the flotation of oxide copper minerals by similar means.

I am not aware that equivalent patents have been issued in the United States. The English patent in question is of a later date than the Schwartz patent above mentioned. The representatives of the Minerals Separation Co. have experimented more or less extensively with this system, while demonstrating their machine to the Inspiration company. As far as I know, they have not proven its practicability. In the course of their experiments, they tried the application of sodium sulphide and sodium polysulphide for this purpose. The latter was produced by treating sulphur with hot caustic soda. At the time these experiments were made, I was not familiar with the chemical action taking place, which, as much as I know now, actually results in the formation of polysulphide mixed with thiosulphates and other oxygen-sulphur compounds. The failure of their experiments, I therefore ascribed to the fact that perhaps a polysulphide, which they were anxious to make, was not actually produced. I proceeded to make sodium polysulphide by a method which I knew, that is, by the treatment of a sodium sulphide solution with sulphur powder. When we applied this reagent to some of our carbonate ores in laboratory flotation experiments, we noted that a good recovery was obtained. The composition of the compound was varied materially in order to find just what composition gives the best results in the flotation of carbonates. Our experience seems to indicate that sodium sulphide alone encourages the flotation of carbonates, but that sodium polysulphide, or sodium sulphide which contains more sulphur than would correspond to the chemical formula Na_2S gives better results. The addition of caustic soda besides the sodium polysulphide was found beneficial.

The question then arose as to why we succeeded in effecting the flotation of oxidized copper when the experiments of the members of the Minerals Separation staff failed. Tests along these lines brought out the fact that the Minerals Separation compound when applied to our carbonate ores also worked successfully, but that it did not on our regular milling ore. Our own compound when added to our mill ore mixture increased the recovery of the carbonates, but evidently interfered with the sulphide extraction, and for this reason seemed to be of as little use as the compound of the Minerals Separation Co. When applying reagents of this character to tailings resulting from ordinary flotation treatment, with the point in view of effecting a sufficient sulphide extraction by the regular flotation process, and using the compound in question only for the purpose of increasing the carbonate extraction, we have found so far that the increase in copper-carbonate recovery over the one obtained without the addition of such chemical compounds is not worth going after.

But this is only a consequence of the fact that carbonates exist in very small amounts only in our milling ore and are partly saved by the ordinary flotation process.

There is no real difficulty about saving carbonates by the method mentioned, if they exist in quantities that make it worth while to save them. That copper carbonates can be recovered may easily be demonstrated by treating a deslimed feed in a series flotation machine. If at the point of the machine, where the sulphide recovery is nearly finished, sodium sulphide is added, the decidedly green color of the concentrates in the following compartments leaves no doubt on this point. The desliming of the feed seems to assist in the carbonate recovery.

It is of considerable (even if only theoretical) value to establish why sodium sulphide and polysulphide tend to increase the recovery of copper carbonates. A coating that might be expected to form cannot be detected. The concentrate resulting from the treatment of pure carbonate ore is decidedly green; besides, when an alkaline condition of the pulp is used there is very little, if any, tendency for any sulphide coating to form, and the alkaline state of the pulp is (as explained above) exactly the condition under which the best carbonate extraction results. Another point that seems to contradict the explanation of these results by the assumption of a sulphide coating is, that when we proceeded exactly as suggested by Mr. Schwartz—*i.e.*, when the application of soluble sulphide was followed by the addition of flotation agents and by the actual flotation—we seemed to obtain poorer results than when the procedure was reversed by applying the oil first and following with the application of some soluble sulphide, although the latter method would certainly seem less favorable to the formation of a sulphide coating, and perhaps for this reason has not been suggested by Mr. Schwartz.

Another theory that has been mentioned as an explanation of this phenomenon is that colloidal sulphur is formed by the solution of sodium polysulphide in water, which, as is known, is a good flotation agent. For instance, it is pointed out in the United States Patent No. 1,140,865 taken out by Dr. R. F. Bacon of the Mellon Institute in Pittsburgh, that by setting free colloidal sulphur, say by the reaction of a soluble sulphide with sulphur dioxide, good flotation results may be obtained as far as the flotation of sulphides is concerned. To make the process available for the flotation of carbonates and other oxidized copper minerals, he suggests that a sulphide coating be first formed on the minerals, *i.e.*, to follow Mr. Schwartz's idea. Whether the colloidal sulphur by itself has a beneficial influence on the recovery of the carbonate (as has been suggested in explanation of our observations) seems rather doubtful when it is considered that we have obtained good results in alkaline solutions in which colloidal sulphur does not seem to separate out from polysulphide containing only a limited amount of sulphur such as was used in our tests. The full theoretical explanation of these facts must therefore be left to future investigations.

Recovery of Silicates

In our experiments with the object of saving the oxidized copper minerals, we soon found that we could save some of these minerals, while others were entirely refractory to the method above mentioned. To establish which minerals could be saved and which not, we attempted an analytical separation into carbonates and silicates. The chemical methods which we tried for the purpose of distinguishing between the two proved unreliable, however, and we had to resort to the separation by specific gravity (panning). The carbonates of copper (malachite and azurite) are heavier than gangue, and the silicates (chrysocolla) are lighter. The separation is rather difficult, owing to the small difference in specific gravity, and the results are therefore far from being altogether reliable, but they seem accurate enough to indicate that the method of saving carbonate copper above referred to is of value only for the recovery of carbonates and does not apply to silicates. This fact seems to be another corroboration of the assumption made above, that carbonates of copper do not float simply because of the formation of a thin surface coating of copper sulphide. It can easily be verified in the laboratory that silicates can be coated with copper sulphide fully as easily as copper carbonates. For this reason, if the filming theory is right, it should be possible to float silicates just as well as carbonates. There is no doubt that they can be floated by transformation into sulphides, only this transformation must not be confined to the surface, but must go deeper. Our experience is, that to effect a good recovery, it is necessary to acidify the pulp so strongly that practically all of the silicates of copper are dissolved and by the action of hydrogen sulphide or other soluble sulphides are transformed into the state of chemically precipitated copper sulphide.

In this form there is no difficulty about the recovery of the copper by flotation, but this procedure is not entirely without objection.

In case hydrogen sulphide gas is used, the acid combined with copper is regenerated. This tends toward a low acid consumption and a good copper extraction, on account of the fact that the treatment winds up with a small percentage of copper in solution and free acid present, both of which are desirable in the light of the law of chemical mass action. But hydrogen sulphide is not a desirable reagent. The fact that it is a gas and not a liquid introduces complications in the apparatus which are accentuated by the fact that it is poisonous and obnoxious otherwise.

Other soluble sulphides used in place of hydrogen sulphide will neutralize some sulphuric acid with the result that the acid consumption will be higher and the copper extraction lower than in case of hydrogen sulphide gas.

As far as acid consumption is concerned, it is pointed out that the free acid lost with the pulp may be settled out in ponds and re-used. However, the re-use of acid diluted to such an extent is a more serious problem than is generally realized.

The treatment of concentrates that are "colloidal," to a much greater extent than ores which millmen have been in the habit of calling colloidal, offers additional problems, which, however, may prove not to be as serious as they look.

Everything considered, I cannot see that the flotation treatment of oxidized copper ores after previously leaching them offers better prospects than straight leaching by decantation and precipitation by other methods.

General Theory

Very much has been published recently about the theory of the flotation process, and very many suggestions have been made that will probably prove valuable after it has been shown by critical tests how far they explain the facts.

It seems to me that an explanation of the qualities of the flotation oils is not as difficult as it might appear. The problem only seems so complicated because the flotation qualities of an oil or an oil mixture have not been separated into their components. In fact, it requires a combination of qualities to make a successful flotation oil. In the first place, the flotation oil has to coat the mineral particles. That there is a tendency for the formation of such a coating can easily be seen from simple experiments. For instance, if samples of copper sulphide (chalcocite), copper carbonate (malachite) and gangue (silica) of the same screen size are spread out on watch glasses and then moistened with a drop of coal-tar creosote, it will be seen that the drop of creosote soon disappears through absorption by the copper sulphide, while it takes a much longer time for it to be absorbed by the copper carbonate and a still longer time with the gangue. On the other hand, when a drop of water is placed on the same minerals, it will disappear on the gangue first, later on the carbonate, and finally on the copper sulphide. This evidently proves that in a mixture of water and oil, the oil will attach itself with preference to the sulphide particles while the water will have the greater tendency to wet the gangue.

The second quality which at least is sometimes required of a flotation oil, is that it has to form a stable froth. In such a case, the stability may be secured by more firmly cementing together the mineral, air and oil. To accomplish this, oils are used which have a tendency to float finely divided gangue particles. The action is characteristic of the heavier pine distillates like pine tar and the lighter ones like turpentine if they are crude, unrefined products; in other words, when they contain some of the heavier distillates. I am not quite sure, however, whether the beneficial influence of oils of this group is not perhaps rather due to the fact that they remove colloidal material from the pulp and thereby improve its tendency to float minerals.

A third quality demanded of a successful oil mixture is that it must be able to produce a sufficient volume of froth. This property is exemplified best by oils of the soluble type—cresol, pine oil, alcohols and other substances. It can easily be proven that when oils of this type are used, although they may be considered insoluble, the water acquires the frothing qualities of the oil. It may be demonstrated by shaking an oil of this character with water and permitting the oil to separate out again. It will be found that the water has acquired frothing qualities by undergoing this treatment. It is perhaps even likely that the soluble portion of the oils belonging to this group is the only portion that is active in this manner. The difference between the oils of group 1 and group 3 may be studied, for instance in a flotation machine of our type. It will be noted that the heavier mineral runs over the concentrate discharge largely in the first compartments forming a heavy, dark froth and the heavy insoluble portions of the flotation oil mixtures apparently go with it. Toward the tailings end of the flotation machines, most of this dark material has disappeared and the froth is lighter and of a more watery nature. The pulp, however, has not lost the quality of forming froth even after it gets to the last compartment of the flotation machines. This permits the conclusion that the frothing characteristics follow the tailings pulp. The water settled out in tanks and tailing ponds has decided frothing qualities. Such water behaves in a similar way to certain alcoholic solutions with which we are used to associate this characteristic, for instance, beer or champagne. The experience of mills using the flotation process, that when the tailings water is reclaimed the quantity of frothing oil may be considerably reduced, further supports the assumption that the formation of froth is caused by water-soluble substances.

Just how the surface tension of water and air must be modified to permit the formation of froth has been made the subject of some speculations recently published by different authors. We have also devoted some thought and a few experiments in our laboratory to this question. A discussion of these matters belongs to the realm of physics, however, and is outside the scope of this paper. But I might add this remark to the discussion of the subject of flotation oils, that most flotation oils not only have the characteristics of one group, but may at the same time possess those of another one. For instance, coal tar has qualities 1 and 3. For the flotation of our ordinary milling ore we do not require much of the quality of oil classified in group No. 2, and, therefore, can get along with coal tar alone. But we find it advisable to add to the coal tar more of the qualities characteristic of the third group, and for this purpose, we add about 5 per cent. of the total in the form of crude pine oil. In many cases it is found that the flotation oil has the characteristics of group No. 2 to such an extent that it is impossible to make clean concentrates. Various chemical means, such as the addition of acid or alkali, are used to counteract this.

CONCLUSION

In summing up I want to say that the fact that the Inspiration company has been able to design a commercially successful flotation plant and has found ways that hold out prospects of raising the plant to a very high state of efficiency, must be attributed to the policy followed by the company of spending great sums of money for the purpose of investigating the flotation process on a commercial scale. In carrying out these investigations, a close coöperation between laboratory and operating force helped us, I believe, more than anything else. I would like to give credit to each person who had a share in contributing toward the success of the work, but cannot do it, because the list would be too long.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 39 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

A New Flotation Oil

BY MAXWELL ADAMS, RENO, NEV.

(Arizona Meeting, September, 1916)

Considerable interest has recently been developed in sage-brush oil because of its possible utilization as a flotation agent in the mining industry. A list of some of its physical properties, together with the method used in its extraction, may prove of interest at this time.

Something over a year ago, a study of the essential oils in desert plants was begun in the Chemical Laboratory of the University of Nevada. None of the oils so far studied possess properties of special interest to engineers, except the oil of sage, *Artemesia tridentate*, which has exceptional power as a flotation agent. This plant, known as common sage brush, also called black sage, is widely distributed over the semi-arid West, being found quite generally on most of the dry plains and mountains west of Missouri.

The method of extracting the oil followed in these experiments is very simple. The leaves, twigs and small branches are placed in an air-tight drum, having a capacity of about 27 cu. ft. Steam is admitted through a number of small openings at the bottom of the retort, and the pressure maintained at 20 to 25 lb. per sq. in. for 3 hr. The escape of the steam from the retort is regulated by allowing it to pass through a stop-cock into a condenser. The water in the receiver is drawn off from time to time and the oil, which is insoluble and floats upon the water, is thus collected. At the end of 2 hr. most of the oil has been driven out, though traces continue to come over for a much longer time. By raising the pressure, the time required could probably be shortened and the yield increased, but the lack of laboratory equipment has prevented the carrying out of this experiment.

The stock wood, bark and branches contain no oil, the distribution of the oil being limited to the leaves and young shoots. There is a seasonal variation in the amount of oil contained. Samples collected on different dates gave the following amount of oil: May 1, 0.42 per cent.; May 27, 0.6 per cent.; June 30, 0.72 per cent.; Aug. 1, 0.9 per cent.; Sept. 10, 1.0 per cent. The increase appears fairly constant from early spring, when the leaves first appear, until light frosts occur in the autumn. When the plant is air-dried there is some loss of oil, as the following data will show: Two 100-lb. samples were collected at the same time. One was distilled when green; the other was air dried for 10 days before distillation. The green sample yielded 275 grams, and the dried sample 248 grams of oil, showing a loss of about 10 per cent.

A laboratory experiment can furnish little data useful in forming an estimate of the commercial cost of production. A man working for 6 hr., and using a pair of common pruning shears, collected twigs which yielded 1 lb. of oil. Since only a small percentage of the oil is lost if the brush is dried, the most economical method of production would perhaps be to collect it in large quantities, by using a tractor engine and a drag, in some such way as land is cleared for farming. When the brush is dry, the leaves and young shoots are easily shaken from the limbs. Thus the amount of material to be distilled would be greatly diminished and the oil perhaps obtained at a cost and in quantity sufficient to make it available as a flotation oil, if not alone, possibly as an ingredient, to increase the flotative power of other oils.

The crude oil is dark in color. When redistilled with steam it is water-white at first, changing gradually to a straw-yellow color upon standing. It has the following physical properties: Density at 15°C., 0.9206. Refractive index at 20°C., 1.4732. Rotation at 20°C., −4.69. At 98°C., a light oil, with a very sharp and pungent odor, begins to distill, but only after the temperature is above 165°C. does rapid distillation take place. At 180°C., the oil turns dark and decomposition begins. At a pressure of 12 mm., and below 125°C. almost all the oil can be distilled.

The chemical properties of the oil are as yet undetermined. There are small quantities of alpha and beta pinene. The main part of the oil has a camphor-like odor and taste, but has failed to give the ordinary tests for ketones. The fraction boiling at 175° to 180°C. has some of the properties of ordinary cineol, but is acted upon by metallic sodium, which indicates that the chief ingredient is not cineol. The chemical composition, which has little interest in this connection, will be worked out later. The important question for the engineer is: Can the oil be produced in quantity and at a cost that will make it available for ore flotation?

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A New Source of Flotative Agents

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

THE reagents now used in flotation consist of various acids or salts, which may be either electrolytes or non-electrolytes, dissolved in water and some substance or combination of substances which function as collecting or frothing agents. At times the only dissolved salts present are those naturally occurring in the water used. The general effect of the electrolyte is to greatly sharpen the separation between the gangue and the concentrate. Examples of this are: The use of sulphuric acid with zinc ores; and of sodium carbonate or calcium oxide (lime) with silver-gold ores. Crude pyroligneous acid is also sometimes used when available. Various oxidizing agents, such as permanganates, bichromates, etc., are added in the selective flotation of lead-zinc ores. Many other reagents for performing certain specific functions, either real or imaginary, have been proposed, and a number of them have been tried upon a working scale. The wild orgy of experimentation which is now going on in flotation exceeds even that which followed the introduction of the Washoe process for the treatment of Comstock ores, when, among other things, sage-brush tea and tobacco juice were reagents added to the pans. Out of all this will, of course, eventually come a more or less standard practice for the treatment of each class of ore.

Omitting a discussion of the functions of frothing and collecting agents, the reagents used in flotation for this purpose may be classified under five general heads: (1) Essential oils; (2) fixed or fatty oils; (3) alcohols and their combinations with organic acids; (4) coal tar and its refined products; (5) petroleum and its refined products.

A flotative agent of some kind is required in flotation as now practiced. A single reagent, as, for example, certain of the essential oils, may perform the dual function of frothing and collecting agent, or a mixture of substances may be required.

In this country, the essential oils used have been wood products, and have been almost exclusively confined to the steam-distilled pine oils. At the present time, the supply of these is limited, and the cost almost prohibitive, so that their use has been dispensed with as far as possible. The oils and tars resulting from the destructive distillation of the various species of the coniferæ find a more general use on account of the greater supply and lower cost, although even these have risen in price and the future supply is somewhat problematical if the demand for them in flotation continues to increase at the same rate as in the past.

The fixed or fatty oils, with the exception of oleic acid and crude pyroligneous acid, are scarcely ever used, since they usually give poor results as compared with other flotative agents.

Coal tar is a common and cheap product, but not all coal tars are suitable for use in flotation. Furthermore, the marketing of coal tar is coming more and more into the hands of large distributors, who contract for the output of the various gas plants. The refined products of coal tar are useful flotative agents, but in the case of certain of these, for example, carbolic acid and cresol, the price at present has become prohibitive. In general, crude coal tar yields the best results when mixed with other oils. It is a frequent practice to use a small proportion of pine oil, in order to modify the froth. Fortunately, in many cases these cheaper mixtures yield very satisfactory results, and can now be obtained at a reasonable cost.

It appears that only the crude petroleum oils having an asphaltic base are generally useful in flotation. These are invariably mixed with other oils. The refined products of petroleum are usually not satisfactory, except as constituents of oil mixtures. The one exception to this is perhaps the case of kerosene acid sludge, which is a byproduct of petroleum refining. Kerosene acid sludge from eastern refineries gave unsatisfactory results at Anaconda, but California kerosene acid sludge, resulting from the refining of crude oils having an asphaltic base, is satisfactorily used in conjunction with a small proportion of wood creosote for the treatment of Anaconda copper ores. While California kerosene acid sludge is a satisfactory flotative agent with many ores, the available supply tends to limit its use. Formerly California refiners threw it away, but now little of it is available upon the open market, since practically the whole of the present production is contracted for a long time in advance by pioneer users.

A suitable oil supply, both as regards the character of oil to give the best results and its present and future availability, is a matter of serious consequence to companies operating flotation plants. It was with this in mind that I began the investigation of possible sources of flotative agents, with the particular object of affording relief to mines located in the arid regions of the West, where none of the common flotative agents are locally available, and where transportation costs upon those from the outside are high.

Certain plants and shrubs have the peculiar property of secreting oil in the new growth, during the growing season, and particularly in the leaves. Botanists appear to be uncertain regarding the function of this oil in the metabolism of the plant. Some hold that it is a reserve food supply for the plant, while others believe that it is waste product which the plant fails to throw off. Be this as it may, many plants, shrubs and trees contain oil in the leaves and new growth. A good example of this is the case of the eucalyptus tree, from the leaves of which essential oils, which are used for various purposes, are recovered by steam distillation. The leaves of the variety known as amygdalina, unique as being the tallest tree in the world, have afforded a large proportion of the flotative agent used in the concentration of complex Australian ores. This variety of tree is fortunately very plentiful in close proximity to these ore deposits.

In the great arid and semi-arid mining regions of the West the most common of the few plants and shrubs native to the region are the varieties of sage brush known as mountain sage, pasture sage, wormwood sage, etc.; also, in certain regions, greasewood and other shrubs of a similar nature. It is, therefore, to sage brush that my attention has been directed in the search for flotative agents for concentrating Western ores.

It appears that the various varieties of wild sage were first investigated by certain members of the Department of Agriculture, with regard to the possibility of producing from them by steam distillation essential oils suitable for pharmaceutical use. Steam-distilled oil was prepared by Rabak[1] from the variety *artemisia frigida* in 1905, and during the summers of 1907 and 1908 larger quantities of oil were prepared from specimens of this plant, collected in South Dakota. In 1912, an essential oil prepared by the steam distillation of *Ramona stachyoides* (black sage) from southern California was also reported by Rabak. This oil was said to contain 40 per cent. of camphor. In 1914, Charles E. Burke and Charles C. Scalione[2] gave an account of an investigation of an essential oil which had been prepared from the same shrub by the steam distillation of several hundred pounds of leaves and twigs collected from brush growing near Riverside, Cal. The yield of oil in this case was 0.9 per cent. of the weight of the material used. This yield was somewhat higher than that reported by the Department of Agriculture. This is perhaps accounted for by the fact that the brush was collected later in the season. This oil is reported as having the following composition:

	Per Cent.
Pinene	6
Cineol	30
Dipentine, terpinene, etc.	25
Thujone	8
Camphor	25
Resinous material	5

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[1] Frank Rabak: Wild Volatile-oil Plants and Their Economic Importance, *Bulletin* No. 235, p. 22, *Bureau of Plant Industry.*

[2] Charles E. Burke and Charles C. Scalione: Investigations on Oil of Black Sage, *Journal of Industrial and Engineering Chemistry*, vol. 6, p. 804 (1914).

Camphor was separated from a portion of the oil, thus demonstrating the rather interesting possibility of black sage as a source of borneol camphor. Upon request, Mr. Scalione furnished me with a small sample of the original oil. This oil is clear, with a very slight yellowish tinge, and has an agreeable odor. In fact, so far as appearance and general behavior goes, although the chemical composition is somewhat different, it very much resembles steam-distilled eucalyptus oil from the Australian variety, *amygdalina*. This oil is a good frothing agent, and it yielded quite satisfactory results upon lead and zinc ores in a qualitative way, although the amount available did not permit of thorough investigation. The idea of investigating sage brush and greasewood as possible sources of flotative agents was conceived early in January, but it was not until early in March that the first 100 lb. of sage brush was forwarded to me,

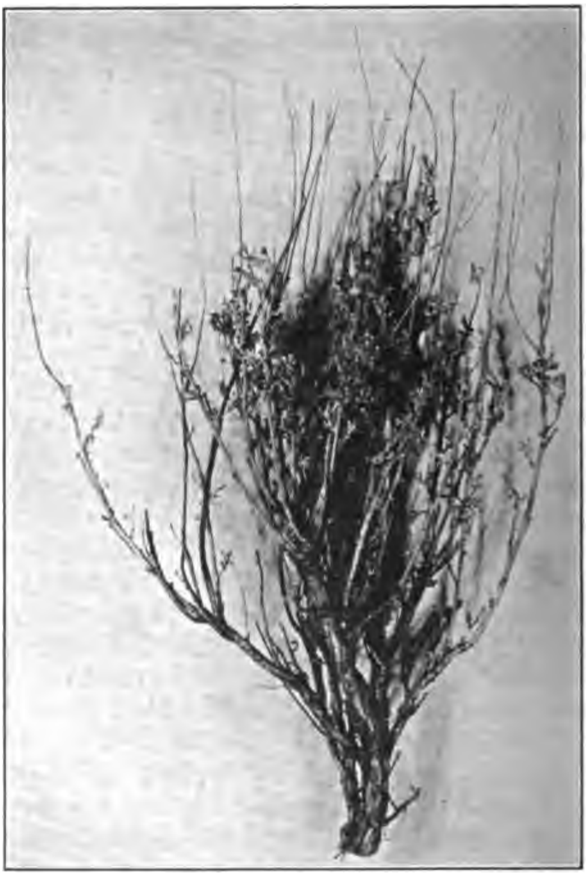

Fig. 1.—Artemisia Tridentata.

through the courtesy of G. B. Lantz. This lot was collected near Goldfield, Nev., and proved to be the variety *Artemisia tridentata.* "In the Western Arid Transition zone the flora consists largely of the true sage brush, Artemisia tridentata,"[3] therefore this would be the variety available in the greatest abundance near to the mines of this region (see Fig. 1).

An apparatus for destructive distillation, capable of treating 30 lb. of brush at a charge, was constructed, and two charges of the brush were distilled (see Fig. 2). The products which first came over consisted of acid liquor resembling the crude pyroligneous acid obtained from wood distillation, a black oil or tar, and inflammable gas. Finally these products ceased to come over, but, upon raising the temperature, a considerable amount of gas was given off and a rather thick reddish-brown liquor, having an alkaline reaction, began to come over, and with it was a small amount of tar similar to that which came over with the acid liquor. The brown liquor had the characteristic fishy odor peculiar to the amines. There was also at times a distinct ammoniacal odor. The acid and alkaline liquors were kept separate, while the tar from both was combined. These three products were first tried qualitatively in the flotation of finely ground samples of various minerals such as galena, cinnabar, pyrite, etc.

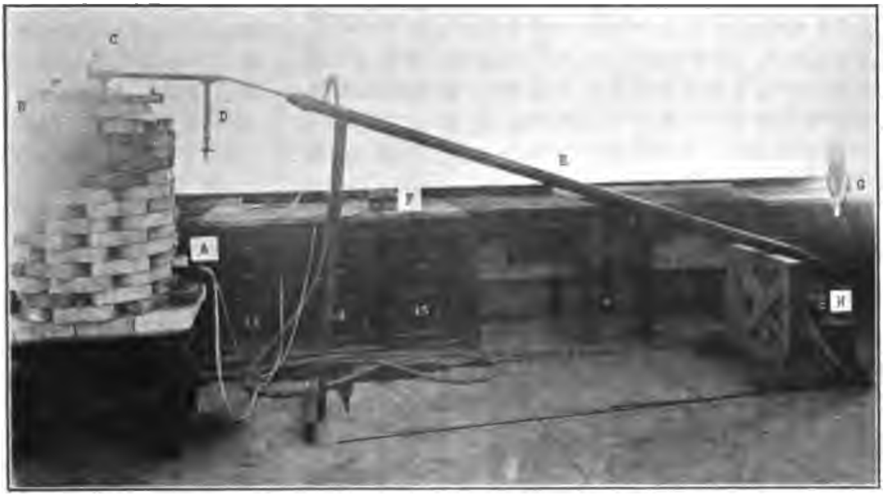

FIG. 2.—EXPERIMENTAL APPARATUS USED FOR THE DESTRUCTIVE DISTILLATION OF SAGE BRUSH. A, burners; B, retort; C, cold junction; D, tar tap; E, condenser; F, galvanometer; G, gas; H, distillate.

The acid liquor behaved very much as does ordinary pyroligneous acid. The alkaline liquor was a good frothing agent, but the froth carried up little mineral. The tar which came over with the acid liquor proved to be a splendid flotative agent.

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[3] *Encyclopædia Britannica,* Eleventh Edition, vol. 27, p. 634.

Following this, quantitative tests were made upon a number of different ores, employing the tar produced from this lot of sage as a flotative agent. These results, a number of which are given below, were in general very satisfactory.

A sample of zinc ore from the Butte Superior mine, Butte, Mont., containing 22.39 per cent. of zinc, when tested in a Janney laboratory machine employing a solution of 0.25 per cent. of sulphuric acid, gave 97.0 per cent. extraction of the zinc. The first concentrate contained 53.8 per cent. of zinc, the second 48.9 per cent., the third 40.5 per cent., and the fourth 18.1 per cent. The oil consumption was at the rate of 0.4 lb. per ton of ore.

A sample of mercury ore from the New Almaden mine, California, containing 0.26 per cent. of mercury, when tested in a Janney machine employing a 0.2 per cent. solution of sodium carbonate, gave an extraction of 90 per cent. of the mercury. The first concentrate contained 3.6 per cent. of mercury, the second 2.5 per cent., the third 1.55 per cent., and the fourth 0.95 per cent. The oil consumption was at the rate of 1 lb. per ton of ore.

A sample of lead ore from the Coeur d'Alene region, containing 12 per cent. of lead, when tested in a Janney machine employing a 0.05 per cent. solution of sodium carbonate, gave an extraction of 92.2 per cent. of the lead. The calculated lead content of the total concentrate was 37.3 per cent. The oil consumption was at the rate of 0.67 lb. per ton of ore.

A sample of silver-gold ore from the Ophir mine, Virginia City, Nev., assaying 0.46 oz. gold and 7.4 oz. silver per ton, when tested in a Janney machine employing a 0.1 per cent. lime or sodium carbonate solution, gave an extraction of approximately 90 per cent. of the silver and 95 per cent. of the gold. The first concentrate assayed gold 5.3 oz. and silver 198 oz. per ton; the second, gold 3.75 oz. and silver 72.9 oz. per ton; and the third, gold 1.32 oz. and silver 35.3 oz. per ton. The oil consumption was at the rate of approximately 0.6 lb. per ton.

The oil consumption when employing sage tar appears to be less than with most of the other oils experimented with in treating the same ores, and the extraction was in general better. Since it is the experience of many that large-scale operation requires less oil than is indicated by small-scale tests, it is reasonable to suppose that the oil consumption in treating the ores cited would be materially lessened when working upon an operating scale, and that possibly the extraction could be bettered. I think the latter is particularly true of the Ophir ore.

Later in March, another lot of brush of the same variety was collected by Mr. Lantz from the same locality, more fully in leaf than that collected earlier in the season. Distillations were run on this lot, keeping the various products separate, so that they could be properly measured and weighed.

Temperature measurements were made at regular intervals by means of a thermocouple, and as a result there was better control of the heating than in former tests. The yield from the last of these tests, which was the most carefully conducted, and was therefore the most representative, was as follows:

	Pounds	Per Cent.
Weight of sage brush used	30.	
Acid liquor	8.4	28.00
Tar with acid liquor	1.1	3.66
Alkaline liquor	0.66	2.20
Tar with alkaline liquor	0.09	0.30
Charcoal	10.00	33.33
Gas (by difference)	9.7	32.51

The retort was slowly heated for 1 hr. before liquid began to condense. The temperature at the center of the charge at the end of this time, as indicated by the thermocouple, was 60° C.; the temperature at the sides of the charge was probably somewhat higher.

In the next period of 6 hr., during which the acid liquor and most of the tar came over, the temperature rose from 60° to 275° C.; the rise above 100° C. taking place during the last hour and a half.

The alkaline liquor and the last of the tar came over in the last period of 3 hr., during which the temperature rose from 275° to 611° C.

It is reasonable to assume that a yield of about 4 per cent. of the tar oil can be realized if the sage is collected at the proper season and the distillation carried on by the best methods. Then there is the acid liquor, which in certain cases could be used directly in flotation; or it might prove profitable to recover the alcohol, acetic acid, the dissolved tar oil, etc., which it contains. I have not had an opportunity to investigate the alkaline liquor thoroughly, but it would seem to present many interesting possibilities. Among other things, the first lot was found to contain 2.09 per cent., and the second lot 2.18 per cent. of nitrogen.[4] I suspect that this liquor may also contain phenolic bodies which would be useful in flotation when separated from the other constituents. The charcoal is fine, but it should be possible to utilize it for fuel in heating the retorts. One peculiarity is the high percentage of ash which it contains (10.5 per cent. in the one sample analyzed). This may be in part due to dust upon the sage brush, although the brush was chopped fine before distillation, and it would seem that a good deal of the dust would be shaken from it during this operation. If the charcoal were burned, as previously suggested, the alkaline ash might serve instead of lime or sodium carbonate in cases where flotation in alkaline solution was practiced.

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[4] Analysis by Professor R. E. Swain.

The inflammable gas would, of course, be burned under the retorts. The proportion of the heat necessary for carrying on the operation which could be realized in this way is problematical; however, it is reasonable to assume that a considerable part of that required for destructive distillation could be produced by burning the gas and charcoal. Moreover, another interesting possibility in connection with the heating of the retorts is the utilization of the waste heat from various metallurgical operations.

Through the courtesy of Charles Scalione, several ounces of steam-distilled essential oil was prepared from the tip ends of a portion of the last lot of sage brush. Approximately 1 hr. was required for distilling each charge of 20 lb. The yield amounted to 0.43 per cent. of the whole plant. This oil had a greenish-yellow color when first distilled, but became yellow upon standing. It has the characteristic penetrating odor of sage, and a very decided tendency to creep up the sides of the glass containing vessel. This oil appears to have distinctly different properties from the essential oil resulting from the steam distillation of the black sage. If gives promising results with some ores, but in my opinion it is not nearly so good as the tar oil resulting from the destructive distillation of the same shrub. In fact, under ordinary circumstances, I do not think that the steam-distilled oil can be given serious consideration, since the mines that would be most benefited are generally located where fuel is high, and the yield of oil is rather small, probably less than 1 per cent. In addition, there are no other valuable products, unless the tannin extract resulting during steam distillation should have a market value or a demand should arise for "sage tea."[5]

Time has not been available for making analyses and a study of all the products resulting from the destructive distillation of sage. However, sufficient work has been done to show at least that the light tar is an efficient flotative agent for a considerable number of ores, as indicated by the tests cited.

The idea of utilizing sage brush in metallurgy is by no means new, as is shown by the following quotation,[6] referring to early development of the Washoe process, and the wild riot of experimentation accompanying it:

"The native sage brush, which everywhere covered the hills, being the bitterest, most unsavory, and nauseating shrub to be found in any part of the world, it was not long before a genius in charge of a mill conceived the idea of making a tea of this and putting it into his pans. Soon, the wonders performed by the sage-brush process, as it was called, were being heralded through the land."

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[5] Patents are pending covering the use of the various sage products in flotation.

[6] Dan De Quille (William Wright): *The Big Bonanza*, pp. 138, 140, American Publishing Co., Hartford, Conn., 1877.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Antecedent Mineral Discovery Requirement

BY E. D. GARDNER, M. E., MISSOULA, MONT.

(Arizona meeting, September, 1916)

APPARENTLY the widespread agitation for the codification of our mining laws has had its effect, and it is quite possible that Congress will take up the question during this present session. The greatest objection to our present statutes pertaining to metalliferous deposits seems to be directed against the law of the apex. Those who are not in favor of changing this law are very far in the minority and they apparently realize that it is useless to urge its retention.

Next in importance to the repeal of the law of the apex, seems to be a widespread advocacy for the repeal of the law requiring discovery of mineral before a valid location of a mining claim can be made. To a lesser extent there appears to be a desire to have the law changed so as to allow patent for claims without mineral discoveries.

The purpose of this article is to review the arguments in favor of the proposed change of the discovery requirement, to show some of the evils that may be expected to result therefrom, and to consider whether a system may be devised to correct the undesirable features of the present law in this respect and at the same time not expose the mining fraternity and the general public to the evils which would result from the repeal of this fundamental requirement.

Under the law as at present framed, the validity of a mining location is dependent upon the fact of discovery of mineral within the boundaries of the land claimed.

> "* * * but no location of a mining claim shall be made until the discovery of the vein or lode within the limits of the claim located" (Sec. 2,320, U. S. Rev. Stat.).

All authorities agree that discovery is the source of the miners' title. Lindley, in his valuable work on Mines (3d Ed., Sec. 335), lays down the rule in part as follows:

> "Discovery is the initial fact. Without that no right can be acquired. * * * Such discovery must precede the location, or be in advance of intervening rights. The proof of recording and marking a claim will not authorize the court to presume a discovery."

As is well stated by the U. S. Supreme Court in Erhard v. Boaro (113 U. S. 536):

> "A mere posting of a notice on a ridge of rocks cropping out of the earth or on other ground, that the poster has located thereon a mining claim, without any discovery or knowledge on his part of the existence of metal there, or in its immediate vicinity, would be justly treated as a mere speculative proceeding, and would not of itself initiate any right. There must be something beyond a mere guess on the part of the miner to authorize him to make a location which will exclude others from the ground, such as the discovery of the presence of the precious minerals in it, or in such proximity to it as to justify a reasonable belief in their existence."

Accordingly, if the miner has made a discovery and otherwise conformed to the requirements of the law, his possession of the claim located is exclusive, against all the world, even, in all probability, against the owner of the land, the Government, except on areas which Congress by special legislation has set apart and defined for paramount purposes, such as National Parks. He has an interest in the land which descends to his heirs, which he can dispose of by will, by deed, or other contract, and all that is required of him after location, until patent is secured from the Government, is to keep his claim alive by performance of the annual assessment work thereon.

In cases where two private locators claim the same ground, the one first making a discovery of mineral would probably get the area, irrespective of the relative time of beginning work or of posting notices. However, such cases are extremely rare, considering the total number of claims located each year. Strictly speaking, a prospector cannot make a valid location before he has discovered mineral, but he has the rights of possession, except against the United States; and, while there is a conflict in the decisions on this point, in many cases the rule appears to be that as long as he is working the ground and is in possession the law will protect him, and when a discovery is made his location is good.

In any attempt to remedy defects in the present system of Federal mining laws, the underlying intent and purpose of existing legislation on the subject should not be lost from view. As was well said by the Assistant Secretary of the Interior in Cataract Gold Mining Co. *et al.* (43 L. D., 248):

> "The intent of the general mining laws was to encourage and promote the development of the mining resources of the United States."

Proposed amendments of existing law, not based on this broad policy but having in view the exploitation of the public domain in the interests of mere individuals without corresponding benefit to the people as a whole, should meet the condemnation of every honest and right-thinking man. Keeping in view, therefore, this beneficent purpose of the present law, let us consider whether the proposed elimination of the antecedent-discovery requirement from the law will operate to give greater effect to such purpose.

It is contended that, if the requirement of an antecedent discovery of mineral within the lines of the claim located were dispensed with, the result would be to hasten the development of the mineral resources of the public domain and so increase the wealth of the nation; furthermore, that the present discovery requirement is an unnecessary hardship in a great many cases, and should be dispensed with in justice to those who are honestly endeavoring to develop the mineral resources of the public domain.

It is urged that, if the antecedent-discovery requirement were abolished, the one first staking out a piece of ground would be protected in his possession of it even if a later locator on the same ground were the first to discover mineral.

This raises the question, to whom should the desired reward (viz., exclusive occupation of the ground) be given? Shall it be to him who is diligent in staking out his claim, or to him who is not merely diligent in that respect but, not content with this, pursues diligence to the point of discovering the hidden mineral wealth? Which of the two is more deserving of reward? Which of the two is bending his efforts toward the consummation of the policy of the law? Certainly the one discovering mineral has performed a very essential act in the development of a mine.

In some mining districts where outcrops are few, extensive and expensive development work is necessary before valuable mineral is found. Many long and costly exploratory tunnels have been driven and shafts sunk to develop tracts of land on which occur no surface indications of veins or mineral but which were favorably situated in mineral belts. Justification for prosecution of development work under such conditions depends to a great extent on a careful study of mineral showing on adjacent lands. The successful prosecution of such exploratory work is dependent on the raising of sufficient capital for the enterprise. The raising of capital, a difficult undertaking in connection with almost any project so essentially speculative in character as that of exploring for precious metals, is made more difficult if the promoters cannot show a fee-simple title to the property to be developed. Hence, the abolishment of the discovery requirement would have the effect of overcoming this one obstacle to the development of our mineral resources.

When valuable orebodies are found on unpatented claims, there is always the menace of litigation to prove the ownership of the ground. After a strike has been made, it often happens that abandoned locations, which at one time covered the land, are resurrected or the boundary lines of prior adjoining claims are moved in such a manner as to take in the more valuable ground. The indefinite manner of describing claims in location notices, and the great number of old illegible mining corners usually scattered over the landscape in mining districts, increase the difficulty of successfully defeating such contests. The richer the orebodies found in these cases, the stronger the efforts by contesting claimants to obtain the ground.

The danger of contests on unpatented ground after a strike has been made will be augmented if valid locations can be made without a discovery of mineral. Contests initiated by the locators of abandoned prior claims would be more likely to succeed if these claims had had a legal existence.

It may be well to say here, however, that it must not be taken for granted that all contests are not made in good faith. Locators of adjoining ground quite often do not know the position of each other's lines, and conflicts occur, therefore, when mineral survey is made for one or both of the claims. A contestant is usually honest in his belief in his ownership of the disputed area. Private contests against the issuance of patent for mining claims are being continually initiated by claimants for all or part of the ground.

Sometimes a locator tries to get part of another's prior location by getting his conflicting claim patented. I know of instances where valuable ground which was being held under location has been patented by a subsequent locator before the owner of the prior claim knew what was going on.

If we shut our eyes to actual conditions and assumed that all locators of mining claims act in perfect good faith in staking out ground with intent to develop it for its mineral contents, probably there would be no necessity for the antecedent-discovery requirement. But mining locations, even now, are made to cover a multitude of frauds against the public-land laws. Title to valuable water-power sites, immense quantities of timber and valuable town sites have been secured under the mining laws. How much greater and more frequent would be these frauds if even a perfunctory discovery of mineral were not required to validate the locations. By far the greater number of mining claims desired for purposes other than mining have had no discovery of mineral.

Care should be taken that the interests of all the people are not sacrificed in our consideration of the welfare of the prospectors. The discovery requirement is designed, and operates, as a check on the disposal of lands of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain.

If the validity of the location is to be dependent merely on taking possession of the ground without the antecedent discovery of mineral, what is to prevent the staking of claims near pay ground with intent to hold them for speculation and hold-up purposes, and not for bona fide development of mineral value, to a much greater extent than at present? What is to prevent irresponsible individuals from staking claims which can be held indefinitely by perfunctory assessment work, thereby tying up large areas of valuable ground for the purpose of levying tribute on those desiring to obtain the ground for mining or other purposes? Surely, to abolish the antecedent-discovery requirement is to open the door to fraud and deceit incalculable, and will tend to block actual development of large areas for many years to come.

The abandonment of the mineral-discovery requirement would naturally facilitate the patenting of mining claims; but, however desirable it may be in some cases to expedite the granting of patents, I believe easy patenting, on the whole, will not benefit the mining industry, but will retard it.

Most mining men know of many promising claims and prospects for which patents have been obtained which have been, to all intents and purposes, abandoned. A large proportion of the applications for patent comes from owners who are tired of doing the annual assessment work but still desire to hold the ground. Of such cases, only a small number of claims are ever worked again.

Nearly 10,000 mineral surveys for groups of claims, from one to thirty or more each, have been made in the State of Montana alone, and this number does not include placer claims that have been taken up by legal subdivisions. While patent does not necessarily follow a mineral survey, it does so in nearly all cases, and of the patented claims only a small number have been worked since final certificates were issued. Outside of the Butte district, I will venture to say that less than 5 per cent. of the patented claims in Montana are now being worked, when metals are higher than at any time in the last 50 years.

In such active districts as Butte and the Coeur d'Alenes, where large capital is required in developing properties, the patenting of claims, on the whole, does not discourage new operations, but perhaps helps them. In cases, however, where one dominant company patents the whole surrounding territory, independent operators who would perhaps be willing to take a chance on developing some of this ground, if open, are thereby kept out.

The patenting of the ground seriously handicaps the chances of small camps and new districts for becoming important producers. Numerous cases can be pointed out where the patenting of the claims has seriously retarded the development of camps, or altogether stopped it.

The majority of those who have written about the desirability of amending the present mining laws have done so from the standpoint of the mining operator or prospector. None of the writers on the subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some people would appear to believe. Individual claims on it are allowed only by virtue of the laws of Congress, which prescribe the performance of certain definite requirements, and it does not belong to the first comer solely on his assertion of a claim to any part of it.

Many abuses have been perpetrated by unscrupulous locators of land under the mining laws. Mining claims have been, and are now, located to control springs, water-holes, range privileges, power and reservoir sites, town sites, rights-of-way, summer-residence sites, timber and agricultural land, natural curiosities, and other surface values. While, at the present time, land more valuable for other purposes than for mining cannot be patented under the mining laws unless mineral has been found and the required expenditure made, in the past, before the Government inspected the ground before issuing patent, valuable areas were patented when the ground was in no respect mineral. At the present time, in some parts of the country, public business is seriously interfered with by unscrupulous locators of ground under the mining laws. While the Government is now embarrassed by these hold-up locations (the mining industry at large has no idea how great the number is), the main expense is the examination of such claims and the delay caused by them. If, by amending the present laws, valid locations can be made anywhere on Government land without mineral discovery, no Government or other activity on public land will be safe from extortion and blackmail. Conditions are bad enough now. For instance, in some localities, as soon as a body of Government timber is advertised for sale, the timbered area and ground over which the logs are to be moved to market is promptly located as mining claims, or old claims are revived, and heavy demands made for the use of the surface of the ground. If, by law, claims of this sort can be made valid, it will give unscrupulous individuals a monopoly of the surface and prevent legitimate business over large areas. The owner of a single valid mining claim crossing a right-of-way of a logging road could exact a sum, for the privilege of crossing his ground, large enough to take all the profit from the logging operations. As the cost of operation is directly related to what an operator can pay for the timber, the Government has a direct interest in all such cases. It has been repeatedly held by the courts that a valid location holds against the Government, even if abandoned for years. I know of areas from which the Government was contemplating selling the timber and on which there were as high as 100 abandoned mining locations. In such cases, if these locations had been valid without a mineral discovery, the locators on reasserting their rights to the ground as soon as they found the timber was valuable could have kept the Government from selling it. There is hardly a stream course or canyon in many parts of the West that has not been at some time plastered with locations.

Even if valid locations could be made without a mineral discovery only on lands classified as mineral, it would still affect the Government, for there are bodies of white pine, which are advertised for sale from time to time, on lands in National Forests which have been classified as mineral but not covered by mining locations.

It has been said that one must actually make a discovery of mineral on land within National Forests before he is allowed possession or allowed to prospect it. Such is not the case, however. A prospector or miner is in no way disturbed by Government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas occupied by individuals under what are known as Special Use Permits; when it is included within a tract from which the Government has sold the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products. Unless the claims are actually interfering with the administration of the Forests, no examinations are undertaken until applications for patent are made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

Mining locations within the National Forests which, after an examination by a competent man, are shown to be clearly invalid, may be disregarded as far as the surface is concerned, but no action is taken to dispossess the locator of the ground and no objection whatever is made to his development of the mineral possibilities. In fact, the Assistant Secretary of the Interior has decided (Nichols-Smith case, unreported) that the Department has no jurisdiction over locations and has authority to cancel a claim only after an application for patent has been made. This decision, however, is now before the Secretary for consideration on motion for the exercise by him of his supervisory authority.

The fact that the Government has placed a large part of its domain within the National Forests shows that this land is valuable to the Government for the benefit of the people at large. This fact alone is an argument from the people's standpoint that the requirements for patenting Government land by individuals, at least within the Forests, should not be made easier.

I believe that, while sometimes the law requiring a discovery before a valid location can be made has worked hardships on prospectors, the wrong that could be worked under a law not requiring a discovery would be far greater.

Under the general mining laws, lands are not required to be chiefly valuable for mineral before they can be entered. The Supreme Court of the United States has held in U. S. *vs.* Iron Silver Mining Co. (128 U. S. 673) that the fact that land may incidentally possess advantages other than its valuable mineral deposits will not preclude its disposition under the mining laws.

An act of Congress of Aug. 4, 1892 (24 Stat., 348) provided:

> "That any person authorized to enter lands under the mining laws of the United States may enter lands that are *chiefly* valuable for building stone under the provisions of the law in relation to placer mineral claims."

The point of difference between this act and the general mining law applicable to mineral deposits is that the Act of 1892 requires the land to be chiefly valuable for building stone.

Power or reservoir sites situated on land containing no evidence of mineral deposits have been entered as stone placers, but the applications for patent have been rejected by the General Land Office. If valid mineral locations could be made on such lands without a mineral discovery, the land could be patented as mining claims.

If patent were allowed for Government ground as mining claims without a mineral discovery there would be very few, if any, power projects free for development at the present time, for the ground would be validly held or patented as mining claims. Most of the undeveloped available power sites on public lands have been included in power withdrawals or National Forests. The ground is, therefore, not subject to entry except as mineral locations, and the only way to get a patent to such ground is under the mining laws. Nearly all power projects in the Northwest, where there is any possibility of early development, are on ground covered by mining locations. In some cases the mining claims are valid under the present law, but the majority have no mineral discovery. If the law were changed concerning mineral discoveries, there would be little new development of hydro-electric power without paying tribute to the kind of prospectors who had the foresight to locate mining claims covering the ground. Also Federal control, due to an interest in the site, could be circumvented by first obtaining patent as mining claims to all the land affected. In the Northwest there is a power project capable of developing 20,500 hp. One-half of the land affected by the project is within a National Forest, and therefore the Government under the present procedure exercises control over the development of the site. The power would be developed under a Special Use Permit issued by the Government, which requires certain conditions to be complied with. All of the public land has been located as placer mining claims, and if the claims were valid the Government under the present law would have no control whatever over the development of the power. In 1913, a permit was issued to a company to develop the power, but for some reason it was not able to go ahead with the project. This company, however, acquired the mining locations, and if the claims could be patented the owners could prevent anyone else from developing the power site. Flour gold is found in gravel on all of these claims, but nowhere in paying quantities.

Another project with which I am familiar is capable of developing 331,000 hp. by using storage, or 178,000 hp. by utilizing only the minimum flow of the river. The lower 12 miles of this project, which is capable of developing 260,000 hp. by storage, or 140,000 hp. without, is entirely covered by mining locations. The area is within a mining district and would probably be classified as mineral, but only a few of the locations have mineral discoveries. One side of the river is within a power withdrawal, and the other within a National Forest. In this case, as in the other, a permit was granted to develop the project, but it has not been done. The permittee has acquired the mining claims and he controls the site in so far as his mining claims are valid. If all these mining claims were valid without mineral discoveries, they could be patented and the Government would lose control of the project.

On March 1, 1906, Windfield Doern and two others located the Eagle placer mining claims, alleged to be chiefly valuable for building stone, on the Stanislaus National Forest in California (41 L.D. 655–659). On Oct. 25, 1907, the claim was conveyed to the Stanislaus Electric Power Co., who filed application for patent on April 5, 1908. The entry was contested by the Government. Stone was used from the claim in the construction of a dam on the claim, from which water was conveyed to a plant 15 miles below, where power was generated. It was proposed to build a second plant on the claim and bring water in a conduit from above. The patent was denied by the General Land Office, as it was shown at a hearing that the stone had no value outside of the construction of the dam, and for that purpose had no special value. The land, however, had value as a power site.

On Jan. 2, 1907, H. V. Gates made application for patent under Act of Aug. 4, 1892, for the Excelsior placer, area 140 acres, in northern California. The patent was protested by the Northern California Power Co. At a hearing, it was disclosed that the ground contained a deposit of basalt that had only a local use for building stone, such as the construction of a power dam, and was not as suitable for that purpose as other common rock. The claim covered a reservoir and tunnel site, and the claimant had a permit from the Forest Service to use a part of the claim for power purposes. The General Land Office held that the claim was more valuable for power than for building stone, and the entry was canceled.

On a river in a western State is situated a power site capable of developing 2,490 hp. The dam site is covered by an unpatented lode claim, and above the falls are situated four patented lode claims along the river in a row. Each of the four patented claims contains a valid mineral discovery, but no ore. Before proceeding to patent, the claimants received $5,000 from a power company for the privilege of flooding their ground.

Perhaps more fradulent mining locations have been made to acquire valuable timber than for any other purpose on non-mineral land. If the mineral-discovery requirement is repealed, large timber steals which have been prevented in the past will be successful. Many applications for timbered mining claims have been canceled after the Government has shown that no mineral discoveries have been made. It would not be possible to defeat such cases if the proposed change of the law is made.

Timber on patented mining claims can be disposed of by the owner as he sees fit. Timber on valid unpatented mining claims on proven mineral land outside of National Forests can be cut for certain described purposes as approved by Congress by Act of June 3, 1878 (20 Stat., 88). Timber, however, on an unpatented mining claim within the National Forests cannot be cut for any purpose other than the development of the claim, except by Special Use Permit or purchase from the Government.

It has been a common practice in some localities to locate mining claims on non-mineral timbered areas to give color of title to the ground while cutting the timber. Numerous suits have been instituted by the Government to recover the value of timber cut from such invalid claims.

In the case of Anderson *vs.* United States (152 Fed., 89 to 91), Anderson, Baker and Sandlin located as placer claims 1,200 acres of non-mineral Government land in the Boise land district, Idaho, from which was cut timber worth $17,751.79. Suit was brought by the Government to recover the value of the timber, and it was successful.

In the case of Powers *vs.* United States (119 Fed., 562), the Government brought suit to recover the value of 668,000 ft. b. m. of timber, worth $7,241, which was cut from non-mineral Government ground. The ground was covered by mining locations made by a third party.

About 10 years ago, a company applied for patent for 3,636.99 acres of Government land as placer claims. The magnitude of the case caused examination to be made by the General Land Office, and it was found that on one claim there was a small worked-out placer deposit of gold. There was no possibility of gold occurring in paying quantities on the rest of the group, although fine colors could be found almost anywhere in the country. This land contained an average stand of 15,000 ft. b. m. of white pine and other varieties to the acre, or about a total of 60,000,000 ft. b. m. on the group. The claims were continuous, but excluded and almost surrounded a small area from which the timber had been burned. The entry was protested by the Government and at a hearing held to determine the validity of the group, the claimants defaulted, thereby admitting the truth of the charges. The entry was finally canceled on Jan. 28, 1910. At this time, unpatented fraudulent placer claims were being held in this district which contained a stand of between 200 and 300 million feet of timber. During the summer of 1910, forest fires that destroyed the timber swept over this country, after which all of the timber-mining locations were abandoned.

In 1913, application was made for two groups of claims in Idaho. No mineral had been found on four of the claims of the group which contained a heavy stand of white pine worth $3,800. After a hearing, the entry was held for cancellation by the Commissioner of the General Land Office.

In 1912, application was made for two claims in the Coeur d'Alene district, Idaho. This entry was also held for cancellation by the Commissioner of the General Land Office, after a hearing. These claims were completely surrounded by patented placer claims, but were situated on a small mountain. The ground was covered by a stand of white pine having a stumpage value of $3,478. One of the claimants was interested in a saw mill situated near the claims.

Along in 1906, a mining company (long since defunct) did considerable development work on lode locations in western Montana, and located a large number of claims. The company failed to find any ore, and went out of existence. The first year after the company abandoned the ground, it was located by a man who did no more work on it. He relocated some of the claims for several years, but when no one else showed any desire to obtain the ground, he also abandoned it. In 1914, a lumber company secured timber on the river above these claims. After the lumber company cleared its right-of-way, the mineral locator came along, saw the situation and immediately reasserted his right to some of the ground that controlled the right-of-way. He demanded an excessive sum for the privilege of crossing his ground. His demands were refused and logging operations shut down for about a year. An examination of the mining claims was made by a mining engineer and it was found that the claims were invalid because no mineral had been discovered upon them. After a year's delay, the company constructed the logging road and commenced operations without any further trouble and without paying tribute to the locator. If these locations were valid without mineral discoveries, the logging operations could have been held up indefinitely unless the excessive demands of the locator were met. The claims are in a general mineral belt.

If valid locations of lode claims can be made in the future without a mineral discovery, many town sites are going to be held as mining claims. The following case is now before the Land Department for settlement. In 1909, a lode claim was located in such a manner as to take in all of the level portion of the canyon at the division point of a new railroad in Idaho. On June 27, 1913, application for patent was made. The section in which the claim is situated has been classified by the United

States Geological Survey as non-mineral, and the land is in the Northern Pacific grant. The railroad has a station and some dwelling houses on the ground and part of the claim is also claimed by the heirs of a homesteader. If the mineral locator can prove a valid discovery on the claim, he is entitled to the ground. The railroad began operating about the time the claim was located. Practically a whole town, with a population of several hundred, is on the mining claim, and there is no doubt that the ground is more valuable for town-site purposes than for anything else. A large diabase dike, in which can be found traces of copper, runs through the claim, but there is no indication that paying ore will ever be found upon it. Sufficient development work has been performed. If no discovery of mineral is going to be required, many similar cases may be expected in the future.

The repeal of the mineral-discovery requirement would open the doors for blackmail of all irrigation companies situated like the one in the following case, even when there is not a trace of mineral on the land. In 1910, an application for patent was made for a placer situated in a small park in western Montana. Previous to the mineral application, an application was made to the Secretary of the Interior for the use of the park for a storage-reservoir site, for which purpose the ground is well adapted. It was proposed to impound water during the winter months and high-water periods, to be used for reclaiming about 3,000 acres of arid land. The reservoir permit was approved by the Secretary, but construction has not been begun on account of the placer claim. The Commissioner of the General Land Office decided that sufficient mineral to constitute a discovery had not been made upon the claim and it is now before the Secretary of the Interior on appeal. The park is an old lake bottom and contains sand, clay and gravel to an undetermined depth. This material all contains minute quantities of gold, but in no place, as far as developed, in paying quantities. No serious effort has been made to mine the gravel, and for the last 20 years the ground has been used by the claimant as a ranch. If it is finally decided that the claim contains sufficient mineral for a discovery, the mineral claimant will be legally entitled to the land, and, before the irrigation system can be commenced, the land will either have to be obtained from him by purchase or by condemnation proceedings.

Throughout the country, attempts have been made to obtain possession of numerous springs, mineral or otherwise, by taking up the ground as mining claims. On Aug. 26, 1886, the Margaret Mining Co. made application for the Gray Eagle Lode mining claim (11 L. D., p. 563) in King County, Washington. This claim embraced some springs known as the Hot Springs of Green River which were supposed to have great medicinal value. The entry was canceled by the Land Office because it was shown at a hearing that the land had no value for mineral but was used for a health resort, and that the mining company was formed to acquire title to the Hot Springs.

By the Act of Jan. 31, 1901 (31 Stat., 745) Congress provided that "all unoccupied public lands of the United States containing salt springs or deposits of salt in any form and chiefly valuable therefor are hereby declared to be subject to location and purchase under the provisions of the law relating to placer mining claims." Under this Act many mineral springs having no value whatever for salt, but having supposedly medicinal properties, have been located as placer claims.

On March 20, 1905, Henry Lovely made application for patent for the Lovely placer claim in the Juneau land district in Alaska. It was shown at a hearing that the ground was used as a health resort and water from two springs was conducted to bath houses, the cost of this being applied as mining improvements. It was also shown that the ground was not mineral or saline in character. The claim was canceled Feb. 13, 1907, by the General Land Office (25 L. D., 426).

By decision of April 14, 1913, the Secretary of the Interior held that the Salt Creek placer claims were invalid (21 L. D., 745). These claims were located in February, 1911, under the Act of Jan. 31, 1901, so as to cover a group of hot springs whose waters carried in solution common salt, Epsom salts and Glaubers salts and other chemical substances. As compared to sea water, the waters of these springs were shown to contain about one-eighth as much common salt. It was held by the Secretary of the Interior, from facts determined at a hearing, that the ground had no value for the manufacture of common salt, but possessed distinct curative properties.

In central Montana is situated a mineral spring known as the Appollonaris. The ground at the small side gulch on which the spring is situated was located as a stone placer and the adjoining ground on the main creek as gold placers. No discovery of gold had been made on the gold placers, and the limestone on the stone placer was not valuable on account of its poor quality and the distance from any market or railroad. This spring has a local reputation for medicinal properties and Special Use Permits for erecting bottling works and hotel buildings on the area was desired by several people. The development of the spring was held up for 5 years, but finally the men holding the mining claims decided that they could not patent the ground and abandoned it. The spring is now being developed.

The abrogation of the mineral-discovery requirement would greatly facilitate such fraudulent attempts to obtain ground desired for purposes other than mining.

At numerous points desirable for summer residences on the shore of Lake Pend Oreille, mining locations have been posted, and in many cases there has been a pretense of doing the annual assessment work. In a few cases, application for patent has been made. On some of these claims sufficient mineral has been found to constitute a mineral discovery, but in the majority of cases no mineral has been found.

The Frances Placer, consisting of 20 acres on a bar on the Clearwater River in central Idaho, was applied for under the mining laws. The claimant had farmed the land for at least 10 years previous to making application. The mineral entry was contested and canceled in 1912, and the land was later taken as a homestead.

Mining claims have been patented, and others are being held as locations, on favorably located scenic points along the rim of the Grand Canyon of the Colorado. Two lode claims also have been patented which cross the Bright Angel Trail. Tourists who have gone down this trail and paid their dollar each to the owner of the lode claims can realize his farsightedness in finding his mine just in this particular location. It is of interest to note that no mineral commercially valuable has been found on any of these claims.

On the public domain there are numerous conflicts of mineral claims with homesteads. In all contests, the burden of proof is on the homesteader, as he has to prove the land non-mineral in character. If valid locations can be made without the necessity of a mineral discovery, the homesteader's lot in mountain valleys is going to be still harder to bear, and the good Lord knoweth it is bad enough now.

If the mineral-discovery requirements for the location of mining claims is abrogated, no form of activity can be safely undertaken on unpatented land, and it is difficult to anticipate the many abuses that can be perpetrated.

The uncertainty of ultimate possession of unpatented land in mineral belts could be remedied, I believe, by making provision by law for clearing title to the ground against all except the United States at any time, as is now done at the time of application for patent. If, after development, mineral is found, the ground could then be patented.

Perhaps the method now used in connection with homesteads on the public domain could be adapted to the making of mineral entries. In the case of homesteads on surveyed land, an applicant, at the time of going on the land, makes a filing in the U. S. Land Office, corresponding to the posting of a location notice on a mining claim, and, if no adverse claims are proven and the land is non-mineral, his entry is allowed. This excludes all other private claims to the ground, and, after he has complied with certain prescribed regulations, patent is issued to the homesteader.

If all men were honest, or this country were a Utopia, it would be perfectly feasible to repeal the law requiring the discovery of mineral, but, unfortunately, such is not the case. Under the present law some operators may have been unjustly obliged to defend their rights, but it has been found that laws made to restrain the unjust usually cause suffer-ing to some of the just. The repeal of this law would afford relief to some who are acting in good faith, but at the same time it would let down the bars, allowing an unlimited amount of fraud, and fostering blackmail. That the wrong application, if possible, would be made of any new mining laws, should be expected from the many frauds that have been perpetrated under the present laws. By far the greater number of the mining claims that have been desired for purposes other than mining have had no discoveries of mineral. I believe that by suitable laws it is quite possible to ameliorate the one condition without aggravating the other.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Possibility of Deep Sand Oil and Gas in the Appalachian Geo-Syncline of West Virginia

BY DAVID B. REGER,* A. B., B. S., C. E., MORGANTOWN, W. VA.

(Arizona Meeting, September, 1916)

Introduction

THE exhaustion of oil and gas in the United States is proceeding at a rapid pace. This is especially true in fields where the light oils that furnish the most fuel for internal-combustion engines are found, leading to a late estimate by Arnold that the petroleum resources are 36 per cent. exhausted.[1] The further remark by Dr. David T. Day at the New York meeting of the Institute, in February, 1916, that the operators of the country are now scarcely able to supply the ever-increasing demand for gasoline, leads the writer to believe that a determined effort will soon be made to secure deeper producing horizons in those regions that now furnish high-grade oil, and in which pump stations and pipe lines offer convenient means for the economical handling of new production. A study of Arnold's table shows further that the States that produce paraffin oil exclusively are 45 per cent. exhausted, this figure being much higher than the average of paraffin and asphalt States altogether. The most extensive of all the paraffin-oil-producing areas is the Appalachian Field, extending with the Appalachian Geo-Syncline from western New York southwestward through western Pennsylvania, western West Virginia, Ohio, Kentucky, and Tennessee, where oil of high gravity is always secured. Of these States, Pennsylvania and West Virginia rank highest in the quality of their product. In West Virginia, the oil sands dip to a lower level than in any other locality in the Appalachian Geo-Syncline, the Pittsburgh Coal of the Pennsylvanian being only 50 ft. above sea level along the Nineveh Syncline at Wileyville, Wetzel County.[2] The very significant fact that the sands of the Mississippian and Upper Devonian Measures furnished the richest oil pools of the State along this deep basin, most of them being non-water-bearing, should give pertinency to such information as may be obtained regarding deeper sands in this region, in which there may be a possible duplication of conditions in the more shallow sands.

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* Assistant Geologist, West Virginia Geological Survey.

[1] Ralph Arnold: Petroleum Resources of the United States, *Economic Geology*, vol. 10, p. 710 (December, 1915).

[2] Ray V. Hennen: Marshall-Wetzel-Tyler Report, *West Virginia Geological Survey*, p. 66. 1909.

Authority for Data

Information has been secured primarily from the published reports of the Geological Surveys of West Virginia, Ohio, Pennsylvania, and Kentucky, supplemented by various special reports and maps of the United States Geological Survey. The southwestward extension of the Chestnut Ridge (Warfield, Campton) Anticline through Kentucky has been made on the authority of a recent publication by James H. Gardner.[3] Several important well records used have never been previously published, having been lately secured by the West Virginia Geological Survey. Due credit has been given in the table of well records to the operators who furnished the various logs, in all cases where this information is available. Special acknowledgment is made to Dr. I. C. White, State Geologist of West Virginia, for many valuable suggestions from his extensive knowledge of the region under discussion.

Description of the Appalachian Geo-Syncline

The Appalachian Geo-Syncline, lying west of the Appalachian Mountain System, and being roughly parallel to it, extends in a south-southwestward direction from New York State through Pennsylvania, West Virginia, Kentucky, Tennessee, and Alabama. As partially shown on Plate 1 (in those regions where detailed structure maps are available) it has a course almost straight through Pennsylvania from Brookville, via Kittanning, Pittsburgh, and Washington, to the southwestern corner of the State where it enters West Virginia and extends southwestward across Wetzel, Tyler, and into Ritchie, where the Burning Springs Anticline cuts diagonally across it shifting the line of its axis nearly 20 miles farther west, to a point near Parkersburg. From this point its course is roughly southwest through Wood, Jackson, Mason, Cabell, and Wayne Counties, to the Kentucky State line 7 miles south of Catlettsburg. Its course through Kentucky is apparently interrupted by the Chestnut Ridge (Campton) Anticline which cuts across it, separating it from its southern extension.

Geology of the Appalachian Basin

The surface geology along the axis of the Appalachian Basin is principally that of the Coal Measures, but at the eastern rim these rocks rise rapidly toward the Allegheny and other mountain ranges of the Appalachian System until they disappear above the tops and the rocks of the

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[3] James H. Gardner: A Stratigraphic Disturbance through the Ohio Valley, Running from the Appalachian Plateau in Pennsylvania to the Ozark Mountains in Missouri. *Bulletin of the Geological Society of America*, vol. 26, pp. 477–483 (December 4, 1915).

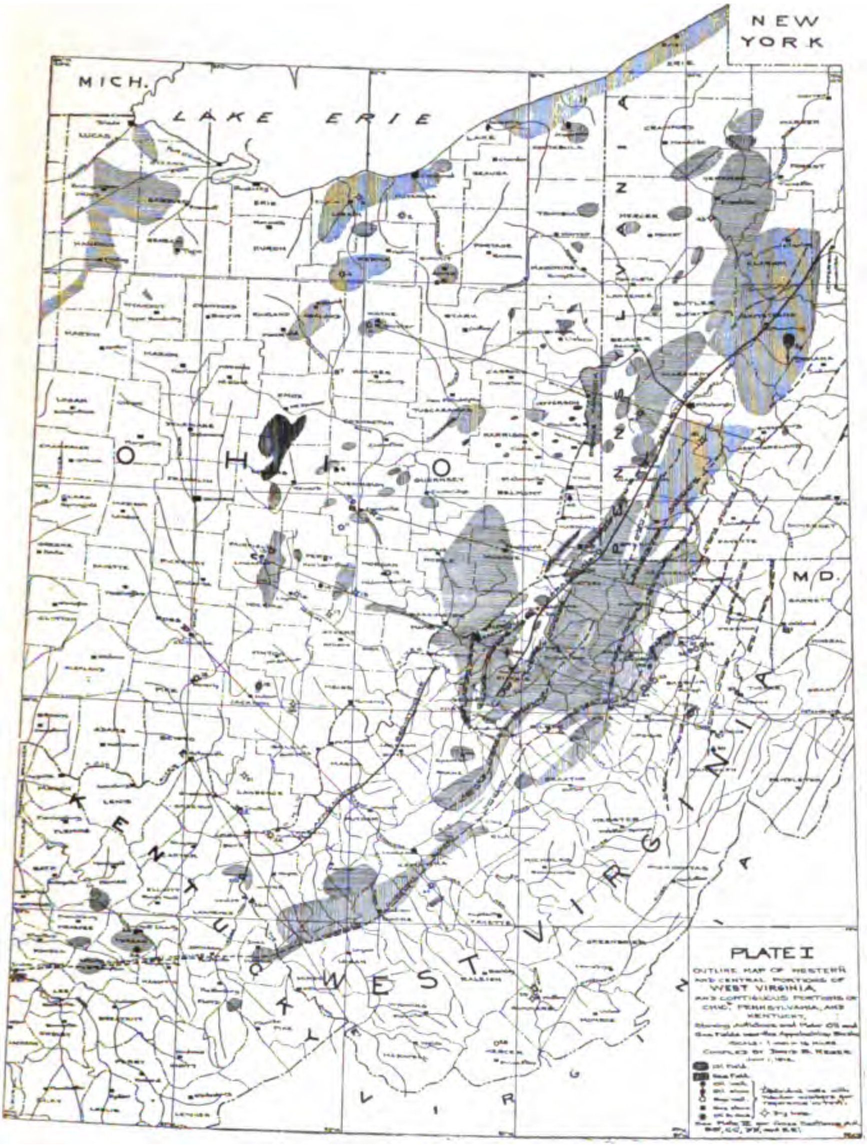

NEW YORK
MICH.
LAKE ERIE
OHIO
PENNSYLVANIA
MD.
KENTUCKY
WEST VIRGINIA
VIRGINIA
PLATE I
OUTLINE MAP OF WESTERN AND CENTRAL PORTIONS OF
WEST VIRGINIA
AND CONTIGUOUS PORTIONS OF OHIO, PENNSYLVANIA, AND KENTUCKY.

PLATE. 1.—OUTLINE MAP SHOWING ANTICLINES AND MAIN OIL AND GAS FIELDS NEAR THE APPALACHIAN BASIN.

Devonian, Silurian and lower formations come to the surface, the pitch in this region being abrupt and irregular. On the western side of the axis the measures rise gradually toward the Cincinnati Anticline, showing none of the spectacular structural disturbances visible on the east. Northeastward from West Virginia, through Pennsylvania, the strata rise gradually along the axis of the basin, but the Pennsylvanian still forms the surface rocks at the New York State line, near Olean. Southwestward from West Virginia through Kentucky, the rocks rise along the axis, but very gradually, so that the Pennsylvanian still makes the surface deposits at the Tennessee line.

Plate 2 shows a columnar section made for the locality of Marion, Harrison, Wetzel, and western Monongahela Counties, West Virginia, where the rocks of the Appalachian Basin dip to their lowest level, and where the oil reservoirs of the State are the most prolific. In this region the surface rocks are principally the transitional Permo-Carboniferous type represented by the Dunkard Series. Throughout this region, numerous drillings are available that extend to the base of the Catskill Series, leaving no uncertainty as to the character of the sediments down to this level. Below the Catskill, the section is based on the deep wells drilled in Pennsylvania (Nos. 41 and 42 on Plate 1) and on numerous drillings in Morgan, Muskingum, and other Counties in Ohio where the Devonian shales have less thickness than in southern Pennsylvania, and on the fairly well-known thickness and character of these Devonian and Ordovician beds in the Alleghany Mountain region of West Virginia, farther east where these measures outcrop. This surrounding information makes the portion of the section below the Catskill more than hypothetical, the probability being that the intervals shown are fairly good while some of the individual formations may be locally absent.

Dunkard Series.—The Dunkard Series, composed of 1,150 ft. of sandy, red, and variegated shales, gray sandstones, with a few thin coals and limestones, is barren of oil and gas in this region and the lower 300 to 400 ft. of the series found below the main drainage channels must be drilled through without result.

Monongahela Series.—The Monongahela Series, composed of 400 ft. of gray sandstones, gray shales, limestones and coal beds, is also practically barren of oil and gas, and is encountered by the drill everywhere along the deeper part of the basin. At its base is the great Pittsburgh Coal bed used widely as a key rock by the drillers of Pennsylvania and West Virginia.

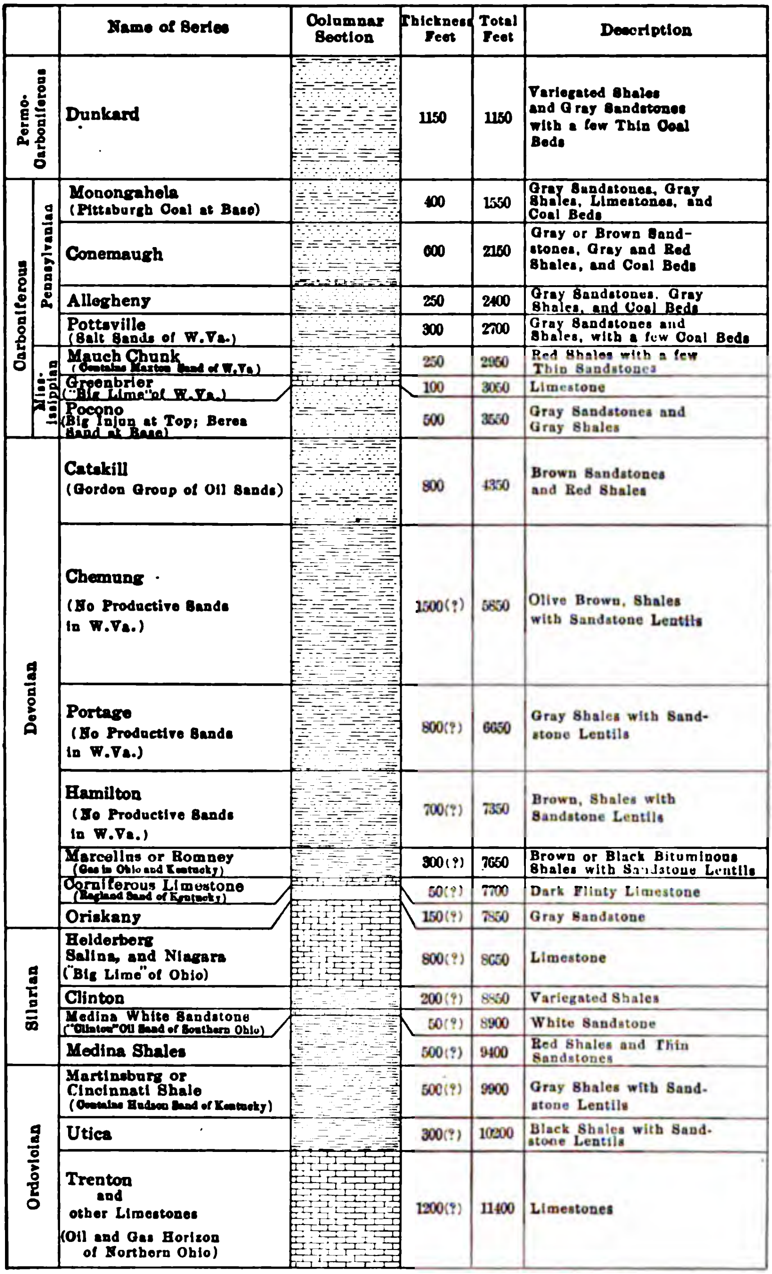
Conemaugh Series.—The Conemaugh Series, comprising 500 to 600 ft. of gray sandstones, red and sandy shales and a few coals and limestones, contains the Little Dunkard sand in the lower portion and the Big Dunkard at its base, both of which have produced oil at several localities in the State.

Allegheny Series.—The Allegheny Series, 250 ft. thick and composed of gray sandstones, gray shales and having a few coals and thin limestones, contains the Burning Springs and Gas sands, that produce oil in great quantity along the Burning Springs Anticline.

COLUMNAR SECTION FOR CENTER OF WEST VIRGINIA OIL FIELDS

(Marion and Surrounding Counties)

Vertical Scale: 9/16″ = 1000 Ft.



		Name of Series	Columnar Section	Thickness Feet	Total Feet	Description
Permo-Carboniferous		**Dunkard**		1150	1150	Variegated Shales and Gray Sandstones with a few Thin Coal Beds
Carboniferous	Pennsylvanian	**Monongahela** (Pittsburgh Coal at Base)		400	1550	Gray Sandstones, Gray Shales, Limestones, and Coal Beds
		Conemaugh		600	2150	Gray or Brown Sandstones, Gray and Red Shales, and Coal Beds
		Allegheny		250	2400	Gray Sandstones, Gray Shales, and Coal Beds
		Pottsville (Salt Sands of W.Va.)		300	2700	Gray Sandstones and Shales, with a few Coal Beds
	Mississippian	**Mauch Chunk** (Contains Maxton Sand of W.Va.)		250	2950	Red Shales with a few Thin Sandstones
		Greenbrier ("Big Lime" of W.Va.)		100	3050	Limestone
		Pocono (Big Injun at Top; Berea Sand at Base)		500	3550	Gray Sandstones and Gray Shales
Devonian		**Catskill** (Gordon Group of Oil Sands)		800	4350	Brown Sandstones and Red Shales
		Chemung (No Productive Sands in W.Va.)		1500(?)	5850	Olive Brown, Shales with Sandstone Lentils
		Portage (No Productive Sands in W.Va.)		800(?)	6650	Gray Shales with Sandstone Lentils
		Hamilton (No Productive Sands in W.Va.)		700(?)	7350	Brown, Shales with Sandstone Lentils
		Marcellus or Romney (Gas in Ohio and Kentucky)		300(?)	7650	Brown or Black Bituminous Shales with Sandstone Lentils
		Corniferous Limestone (Ragland Sand of Kentucky)		50(?)	7700	Dark Flinty Limestone
		Oriskany		150(?)	7850	Gray Sandstone
Silurian		**Helderberg Salina, and Niagara** ("Big Lime" of Ohio)		800(?)	8650	Limestone
		Clinton		200(?)	8850	Variegated Shales
		Medina White Sandstone ("Clinton" Oil Sand of Southern Ohio)		50(?)	8900	White Sandstone
		Medina Shales		500(?)	9400	Red Shales and Thin Sandstones
Ordovician		**Martinsburg or Cincinnati Shale** (Contains Hudson Sand of Kentucky)		500(?)	9900	Gray Shales with Sandstone Lentils
		Utica		300(?)	10200	Black Shales with Sandstone Lentils
		Trenton and other Limestones (Oil and Gas Horizon of Northern Ohio)		1200(?)	11400	Limestones

PLATE 2.—COLUMNAR SECTION FOR CENTER OF WEST VIRGINIA OIL FIELDS.

Pottsville Series.—The Pottsville Series, composed of gray conglomeratic sandstones, gray shales, and having numerous coal beds, varies in thickness from about 200 ft. in the northern panhandle to nearly 4,000 ft. at the southern end next to Virginia. Along the axis of the Appalachian Basin, its thickness at the Pennsylvania line is about 250 ft., and this increases gradually to nearly 1,000 ft. at the Kentucky line. Oil is often found in several of its sands, the upper one being called the Second Cow Run, and the lower ones being usually termed the First, Second, and Third Salt sands. Copious flows of salt water are usually found in one or more of these horizons, giving rise to their names.

Mauch Chunk Series.—The Mauch Chunk Series, composed mainly of red and green shales, with a few flaggy green fine-grained sandstones and impure limestones, varies in thickness from only a few feet in some of the western counties to about 3,500 ft. in Summers and Mercer. Along the oil belt it is usually about 250 ft. thick, with the Maxton oil and gas sand near the base.

Greenbrier Limestone.—The Greenbrier limestone, usually a solid mass of hard limestone, varies in thickness from 50 ft. in the northern panhandle to 1,500 ft. in Greenbrier and other southern Counties. Along the Appalachian Basin, it averages about 100 ft. and is known as the "Big Lime," and is the most dependable key rock below the Pittsburgh Coal, as the intervals from it to the lower sands are fairly constant over wide areas. It is seldom a producing horizon but makes some oil and gas from sandy streaks in Lincoln, Wayne, and Mingo Counties.

Pocono Series.—The Pocono Series, composed of gray sandstones and gray shales, is usually about 500 ft. thick in all the counties along the Appalachian Basin, and contains the Keener, Big Injun, Squaw, Weir, and Berea sands. Of these the Big Injun is the most prolific oil and gas horizon of the State, possibly deriving its oil from the Big Lime which often rests directly upon it; the Berea sand is a close second to the Injun, being of widespread occurrence and often productive over wide areas.

Catskill Series.—The Catskill Series, composed of red or brown sandstones, separated by red shales, varies from 500 to 800 ft. in thickness and contains the Gantz, Fifty-foot, Thirty-foot, Gordon Stray, Gordon, Fourth, Fifth or McDonald, Sixth or Bayard, and Seventh or Elizabeth sands. Of these the Gordon is the most valuable producer, ranking with the Berea in the quantity of oil but not having such a wide occurrence. The sands of this series are the lowest that have been commercially productive in the State. Southward toward Kentucky, the series thins out completely, being scarcely represented at all south of the Great Kanawha River.

Chemung, Portage and Hamilton Series.—The Chemung and Portage, composed of green, gray and brown shales respectively, with lenticular sandstones, the Chemung being probably about 1,500 ft., the Portage about 800, and the Hamilton about 700, contain no productive sands in West Virginia, although the Speechley sand of the Chemung has made shows of oil and gas in various wells. In northern Pennsylvania, there are several productive horizons in this group. Westward in Ohio and southward toward Kentucky, these three series thin out almost completely, leaving the Berea sand only 700 to 1,000 ft. above the Corniferous limestone. These shales form a great barrier that has prevented any exploitation of the lower Devonian sands in northern West Virginia.

Marcellus Shale.—The Marcellus shale, known as the Ohio black shale in the State of that name and as the Chattanooga shale in Tennessee, and often called simply the Devonian black shale, is probably about 300 ft. thick along the Appalachian Basin in West Virginia. It is a black, bituminous horizon, and contains a few lenticular gas sands in Ohio and Kentucky.

Corniferous Limestone.—The Corniferous limestone, probably not more than 50 ft. thick in West Virginia, is the Ragland oil sand of Menefee and Morgan Counties, Kentucky. It has been penetrated in West Virginia by only three wells near the Kentucky line (Nos. 26, 26A, and 27 on Plate 1), and in two or three wells in Randolph and Tucker Counties in the Alleghany Mountains, the detailed records of only one of which (No. 32) has been obtained. It is probable that the most eastern oil counties of the State do not have this horizon, as its presence is not noted in the Alleghany Mountains where it should crop, and where it should be easily recognized from its well-known flints and fossils.

Oriskany Sandstone.—The Oriskany sandstone, gray in color and probably 150 ft. thick, and a producer of oil and gas in southern Indiana and central New York, was found in the Geary Well (No. 42 on Plate 1) in Pennsylvania, where it was 155 ft. thick, and in the Slaughter Creek Well (No. 27 on Plate 1) in Kanawha County, West Virginia, where it was 15 ft., and in the Parsons Well (No. 32 on Plate 1), where it was 80 ft. West of the basin it is not noted in the deep wells of Ohio but on the eastern side of the Alleghany Mountains it is the well-known glass sand of Morgan County, West Virginia, and it seems likely that it is represented along the basin in West Virginia.

Helderberg, Salina, and Niagara.—The Helderberg, Salina, and Niagara limestones, often comprising a solid limestone mass 800 ft. or more thick, and known in Ohio drilling parlance as the "Big Lime," has been penetrated in West Virginia only by the Bartram (No. 26A) and Slaughter Creek (No. 27) Wells, and was found in the Geary (No. 42) Well in Pennsylvania. Since this great limestone mass is found in these three wells and is universally present in Ohio, and is also found in much the same great development in West Virginia east of the Alleghanies, it seems safe to conclude that it is present along the Appalachian Basin with the same or greater thickness. It is possibly the parent reservoir from which much of the White Medina sandstone, or "Clinton Sand" oil of Ohio is derived, so that its presence along the basin is of special interest.

Clinton Shales.—The Clinton variegated shales are usually found in Ohio between the "Big Lime" and the "Clinton" sand, varying in thickness from 50 to 200 ft., and probably occur in West Virginia along the Basin.

Medina White Sandstone.—The "Clinton" oil sand of southern Ohio, believed by I. C. White and others to represent the Medina white sandstone, is the great oil-and gas-producing stratum of Hocking, Fairfield, Perry, and other southern Ohio Counties, being usually 10 to 50 ft. thick. No drill has ever reached it in West Virginia, but it is present in the Alleghany Mountains east of the oil fields and probably occurs along the Basin.

Medina Shales.—The Medina variegated shales, with probable lenticular sandstone horizons, noted in many deep wells in southern Ohio, may have a thickness of 500 ft. along the Appalachian Basin.

Martinsburg Shale.—The Martinsburg shale, often called the Cincinnati or Hudson, is found in Ohio, Kentucky, and eastern West Virginia, and may therefore be taken for granted along the Appalachian Basin in West Virginia. According to Hoeing,[4] this series furnishes several oil and gas sands in Wolfe, Morgan and other Kentucky Counties. Its thickness is possibly 500 to 1,000 ft. along the Appalachian Basin.

Utica Shale.—The Utica formation, consisting of black shales with probable sandstone lentils, may have a thickness of 300 ft. along the basin, although the information regarding it in surrounding States is vague and uncertain.

Trenton Limestone.—The Trenton group, composed of several thick limestones, often coalesced into one thick mass 1,000 to 1,200 ft. thick, is the great oil and gas-producing zone of north-central Ohio and also is productive in Indiana, central New York, and in some parts of Kentucky. The fact that it occurs in these surrounding States and also crops in great thickness in West Virginia east of the Alleghanies seems to leave little doubt that it occurs also, with its usual great thickness, along the Appalachian Basin.

Cross-Sections

Based on the evidence of the numbered wells shown on Plate 1, many of which were very deep tests, and on many hundreds of oil and gas wells, the records of which are published in the West Virginia State Reports, and also on the outcropping rocks around the rim of the Appalachian Basin, five cross-sections, the geographic locations of which are shown

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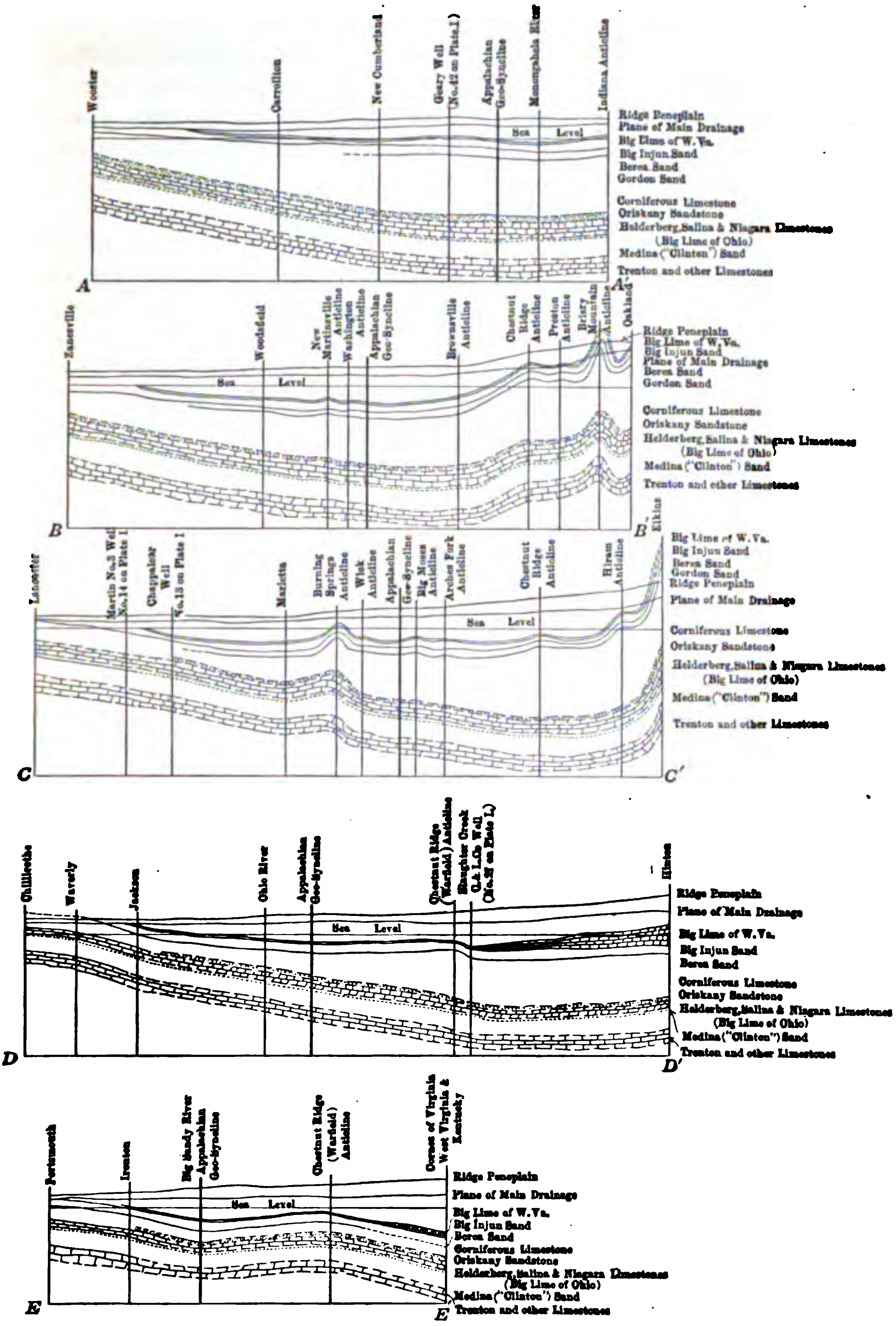
[4] J. B. Hoeing: Oil and Gas Sands of Kentucky, *Bulletin No.* 1, *Kentucky Geological Survey*, pp. 39–40 (1905).



CROSS SECTIONS

Horizontal Scale: $11/32$ = 16 Miles
Vertical Scale: $11/32$ = 4000 Ft.

Solid Lines Indicate Sands Determined from Well Records
Dotted Lines Indicate Supposed Position of Deep Sands as Determined from a few Deep Wells and from Intervals East and West of Appalachian Basin

PLATE 3.—CROSS-SECTIONS AS INDICATED ON PLATE 1.

on Plate 1, have been plotted across the basin in West Virginia (Plate 3). The scale of the plates does not permit the representation in detail of the surface hills and valleys but a solid line shows the ancient peneplain surface of the ridges, and another shows the plane of the main drainage channels with reference to sea level, from which the approximate interval from this latter plane to the principal sands may be determined, as all estimates for drilling must start from this drainage plane. Formations shown with solid lines are plotted from actual records but the gaps are filled in with dotted lines to represent the probable positions of the various deep sands and limestones as outlined above. Of the more shallow sands, only two or three are plotted for use as key rocks. These cross-sections reveal the very interesting probability that the well-known gradual eastward thickening of the Chemung, Portage, and Hamilton shales shifts the axis of the Appalachian Basin of the lower limestones and sands many miles farther east than the position it occupies when referred to the formations above these shales.

Deep-Well History

The accompanying table, containing the abbreviated records of a large number of wells, has been compiled from the various sources indicated to show the different deep tests that have been made in West Virginia and in neighboring States to find oil in the sands below the Catskill Series. In Ohio, where hundreds of productive wells have been drilled to the "Clinton" and Trenton, only those have been selected that lie nearest to the Appalachian Basin, being located along a fairly straight line reaching from Lake Erie to the Ohio River, and in Kentucky only a few wells are listed to show the typical formations and intervals. Each well is given a tabular number that corresponds to the same number placed beside the well symbol on Plate 1.

Of the wells tabulated for West Virginia, only four (Nos. 26, 26A, 27 and 32) were drilled as deep as the Corniferous limestone, and only three of these went through the Oriskany sandstone into the Niagara and Helderberg limestone. No well in the State has ever been drilled into the Medina White sandstone ("Clinton" oil sand of Ohio) or into the Trenton limestone. All the wells drilled to the Oriskany sandstone have been somewhat distant from the main oil fields of the State and are therefore indecisive as tests of its oil and gas possibilities. Briefly, then, the deep sands not yet tested in the main West Virginia oil fields are the Corniferous limestone ("Ragland" of Kentucky), Oriskany, Medina white sandstone ("Clinton" of Ohio), and the Trenton limestone, with a possibility of another sand horizon in the Martinsburg (Hudson) group, which produces oil in Kentucky.

The well most typical of the formation and conditions likely to be encountered in drilling for these deep sands in West Virginia is the Geary deep well (No. 42 on Plate 1) drilled near McDonald, Pa., and only about 50 miles due north of the main West Virginia fields. This well holds the unique distinction of being the deepest hole ever drilled in the United States, and its record down into the Ohio "Big Lime" was published by I. C. White.[5] Judging from the record of this well and that of the Slaughter Creek (No. 27) deep well, it seems likely that troublesome salt water may be expected in the Oriskany and Salina beds along the minor synclines that lie parallel to the main Appalachian Basin, and that oil and gas will more probably be found along the axes of the anticlines or at no great distance down their slopes.

This theory that oil may be found principally along the anticlines is contrary to the experience of operators who have exploited the sands of the Catskill Series in the same region, and who have learned through the drilling of many thousands of wells that these latter sands are usually not water-bearing and the oil is therefore almost universally found along the synclines.

Whether deep-sand oil and gas exist in commercial quantity along the Appalachian Basin in West Virginia will be known only through the ultimate test of the drill, but the fact that these sands are widely productive in Ohio, Kentucky, Indiana, and New York, in regions where structural relief is similar to that in West Virginia, and the further strong probability, and almost certainty, that these porous reservoirs actually exist in close proximity to the great limestones that, along with the Devonian black shales, are evidently their main source of oil in other localities, indicates that they will be found commercially productive along the Basin. The fact that the bituminous shales are undoubtedly much thicker in West Virginia than in Ohio, Kentucky and Tennessee indicates an even more hopeful possibility of oil.

If the doubt as to the presence of oil be dismissed as unreasonable, the problem of the operator resolves itself into the double question; first, as to the location of the pools of oil and gas; and, second, as to the mechanical possibility of drilling wells 5,000 to 10,000 ft. deep. As outlined above, the most favorable localities seem to be along or near the axes of the anticlines where the danger of finding salt water would be minimized and where the gas, if any, should most certainly collect. The search for oil would naturally be made on either side of the anticlines by making locations at increasing distances from the axes.

That drilling may be carried on successfully to the great depths indicated by the figures above seems open to little serious doubt. In 1860, when the search for oil and gas may be said to have been actively begun in the Appalachian Basin, few, if any, wells had been drilled to a

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[5] I. C. White: Notes on a Very Deep Well Near McDonald, Pennsylvania, *Bulletin of the Geological Society of America*, vol. 24, p. 275 (June 10, 1913).

Summarized Record of Deep

No. on Plate 1	Farm Name and Number and Oil Company	County and State	Elevation above Tide, Feet	Big Lime of W. Va.		Berea Sand		Corniferous Limestone		Oriskany Sandstone	
				Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet
1	J. A. Giddings lot, Jefferson O. & G. Co.	Ashtabula, Ohio.	...	...	...	...	...	...	...	...	...
2	Garfield & Caine, East Ohio G. Co.	Cuyahoga, Ohio.	780	...	...	10	80	...	...	...	...
3	Citizens No 1.	Lorain, Ohio.	...	...	...	...	...	...	...	...	...
4	E. S. Albert No. 1.	Medina, Ohio.	920	...	...	386	10	...	...	...	...
5	Barberton Chem. Co. No. 1.	Summit, Ohio.	...	...	...	460	10	2,180	...	...	...
6	Wooster well, H. B. Odenkirk.	Wayne, Ohio.	860	...	...	465	30	...	...	...	...
7	Kaylor.	...	...	...	...	615	10	...	...	...	...
8	Fairall.	Muskingum, Ohio.	...	...	...	834	16	...	...	...	...
9	Dillon Falls well, Chicago-Zanesville O. & G.	Muskingum, Ohio.	...	...	...	895	12	...	...	...	...
10	George Handehsy, Zanesville G. & O. Co.	Muskingum, Ohio.	...	...	...	1,010	23	...	...	...	...
11	W. J. Roberts.	Muskingum, Ohio.	...	250	38	847	14	...	...	...	...
12	McConnellsville Fair Grounds, McConnellsville City.	Morgan, Ohio.	725	625	44	1,279	66	3,082	2	3,125	8
13	T. J. Chappalear, J. P. Fishel.	Morgan, Ohio.	750	...	...	1,150	20	...	...	...	...
14	Martin No. 3. E. S. Martin.	Perry, Ohio.	...	...	...	920	20	...	...	...	...
15	Crouse No. 1. B. S. Stretton.	Fairfield, Ohio.	...	...	...	...	...	...	...	...	...
16	Logan town, Logan town.	Hocking, Ohio.	748	...	...	683	45	...	...	...	...
17	Columbus Hocking C. & I. Columbus Hocking C. & I.	Athens, Ohio.	...	...	...	...	...	...	...	...	...
18	Waverly O. & Gas. Co. Waverly O. & Gas Co.	Pike, Ohio.	575	...	...	...	...	...	...	...	...
19	Buckeye Coal Co. No. 2. Buckeye Coal Co.	Jackson, Ohio.	...	...	...	640	20	...	...	...	...
20	At Vinton village.	Gallia, Ohio.	...	...	...	...	...	...	...	...	...
21	At Mt. Vernon village.	Lawrence, Ohio.	...	...	...	1,095	25	...	...	...	...
22	At Ironton town.	Lawrence, Ohio.	500	...	...	1,010	47	...	...	...	...
23	Burns well.	Morgan, Kentucky,	...	368	106	1,095	10	...	...	...	...

Wells Shown on Plate 1

Helderberg, Salina and Niagara, "Big Lime of Ohio"		Medina ("Clinton") Sand		Trenton Limestone		Total Depth, Feet	Reference to Detailed Record	Remarks
Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet			
1,810	288	2,127	13	...	...	2,140	Ohio G. S. 4th series, *Bull.* 1, p. 303.	Oil well in "Clinton" sand.
1,210	1,455	2,680	...	...	...	2,770	Ohio G. S. 4th series, *Bull.* 12, p. 65.	Heavy flow of water and oil show in Niagara lime. Gas in "Clinton" sand.
440	850	...	...	2,645	...	....	Ohio G. S. 4th series, *Bull.* 12, p. 64.	No "Clinton" sand.
1,512	1,127	2,749	14	...	...	2,777	Ohio G. S. 4th series, *Bull.* 12, p. 62.	
2,180	350	2,500?	30	...	...	3,006	Ohio G. S. 4th series, *Bull.* 1, p. 296.	Gas show(?) in "Clinton" sand. Rock salt and lime in lower 300 ft. of hole.
1,830	1,085	3,104	31	...	...	3,140	Ohio G. S. 4th series, *Bull.* 12, p. 60.	1,440,000 cu. ft. of gas daily and oil show in "Clinton" sand. Heavy flow of brine at 2,055 ft.
1,725	960	2,851	28	...	...	2,957	Ohio G. S. 4th series, *Bull.* 12, p. 59.	Gas show in "Clinton" sand.
2,035	934	3,141	21	...	...	3,194	Ohio G. S. 4th series, *Bull.* 12, p. 37.	"Clinton" sand oil well.
2,120	1,275	3,543	21	...	...	3,567	Ohio G. S. 4th series, *Bull.* 12, p. 35.	Salt water 2,560 ft. and 3,205 ft. Shows of oil and gas in "Clinton" sand.
2,640	1,005	3,709	38	...	...	3,805	Ohio G. S. 4th series, *Bull.* 12, p. 36.	Dry hole.
2,195	970	3,296	...	...	...	3,314	....	Gas in "Clinton" sand.
...	...	...	...	...	...	3,186	Ohio G. S. 4th series, *Bull.* 1, p. 145.	Gas in Berea sand.
2,700	900	3,935	9	...	...	3,947	Ohio G. S. 4th series, *Bull.* 12, p. 38.	No oil or gas; 1 bailer of water per hour in Niagara lime.
2,170	1,170	3,175	17	...	...	3,213	Ohio G. S. 4th series, *Bull.* 12, p. 23.	Oil well in "Clinton" sand.
1,378	...	2,180	5	3,470	...	3,569	Ohio G. S. 4th series, *Bull.* 12, p. 28.	No oil or gas noted in record, but was probably a "Clinton" gasser.
1,715	715	2,627	18	...	...	2,694	Ohio G. S. 4th series, *Bull.* 12, p. 39.	Dry hole.
...	...	...	...	...	...	....	Ohio G. S. 4th series, *Bull.* 12, p. 41.	Drilled to "Clinton" sand. No oil or gas.
485	490	...	...	1,960	808	3,335	Ohio G. S. 4th series, *Bull.* 12, p. 48	Some salt water and gas in Niagara lime; oil and gas show in Trenton.
660	1,454	2,255	8	...	...	2,315	Ohio G. S. 4th series, *Bull.* 12, p. 44.	4,000,000 cu. ft. of gas daily in "Clinton" sand.
...	...	...	...	...	...	3,200	Ohio G. S. 4th series, *Bull.* 12, p. 44.	Dry hole. No "Clinton" sand.
1,810	620	...	...	...	...	2,730	Ohio G. S. 4th series, *Bull.* 12, p. 50.	Dry hole. No "Clinton" sand.
1,827	584	...	...	3,440?	...	3,600	Ohio G. S. 4th series, *Bull.* 12, p. 49.	Dry hole. No "Clinton" sand.
...	...	1,408	94	...	...	1,508	Ky. G. S. *Bull.* 1, p. 159.	"Clinton" sand oil well.

No. on Plate 1	Farm Name and Number and Oil Company	County and State	Elevation above Tide, Feet	Big Lime of W. Va.		Berea Sand		Corniferous Limestone		Oriskany Sandstone	
				Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet
24	Caney Creek well, G. M. Sullivan.	Morgan, Kentucky	...	453	69	1,035	24	1,370	...	...	...
25	Blaine Creek well.	Lawrence, Kentucky	...	...	...	1,660	2	2,592	−5	...	...
26	Central City well, T. H. Harvey.	Cabell, W. Va.	530	970	150	1,730	25	2,760	10	...	...
26 A	David Bartram No. 483, United Fuel Gas Co.	Wayne, W. Va.	595	1,220	80	1,875	20	2,760	410	...	...
27	Slaughter Creek C. & L. Co. W. S. Edwards Oil Co.	Kanawha, W. Va.	640	1,453	207	2,093	12	4,945	90	5,035	15
28	Griffith No. 1, Rich Creek O. & G. Co.	Mercer, W. Va.	2,090	...	...	...	...	...	...	...	...
29	B. T. Baker No. 1, Summers O. & G. Co.	Summers, W. Va.	1,586	1,020	1,057	2,600	25	...	...	...	...
30	P. C. Daniels No. 1. Tygarts Valley Oil Co.	Randolph W. Va.	1,990	...	...	...	...	...	...	...	...
31	Alonzo W. Murphy No. 1. Mead *et al.*	Randolph, W. Va.	1,995	...	...	...	...	...	...	...	...
32	Parsons Pulp & Lumber Co. Parsons Pulp & Lumber Co.	Tucker, W. Va.	1,655	...	...	...	...	3,825	20	3,980	80
33	J. C. Benson No. 3,612 Hope Natural Gas Co.	Barbour, W. Va.	1,026	1,235	90	1,615	138	...	...	...	...
34	Porter Maxwell No. 3,869, Hope Natural Gas Co.	Harrison, W. Va.	1,050	1,531	...	...	...	...	...	...	...
35	M. J. Coplin No. 1, Tri-State Gas Co.	Harrison, W. Va.	1,030	1,288	77	1,711	13	......	...	...	...
36	Absalom Knotts No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,460	960	50	1,460	120	...	...	...	...
37	R. S. Hauser No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,480	950	40	...	...	...	...	...	...
38	Russell McCartney No. 1, Greensboro Gas Co.	Taylor, W. Va.	1,580	1,055	35	1,500	90	...	...	...	...
38 A	T. E. Dye No. 1, Am. Hydroscope Co.	Ritchie, W. Va.	...	...	...	...	...	...	...	...	...
39	J. F. Dobbs No. 1, South Penn Oil Co.	Marshall, W. Va.	980	1,760	30	...	...	...	...	...	...
40	Boggs Run well, Wheeling Dev. Co.	Marshall, W. Va.	900	...	...	...	...	...	...	...	...
41	Wm. Bedell, Forest Oil Co.	Allegheny Penna.	...	1,050	50	1,515	50	...	...	...	...
42	R. A. Geary No. 1,770, Peoples Nat. Gas Co.	Washington, Penna.	1,050	953	29	1,610	12	6,008	37	6,045	155
43	Witherop, Conway Bros.	Venango, Penna.	955	...	...	...	...	3,870	10	...	...

Helderberg, Salina and Niagara, "Big Lime of Ohio"		Medina ("Clinton") Sand		Trenton Limestone		Total Depth, Feet	Reference to Detailed Record	Remarks
Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet	Depth Top, Feet	Thickness, Feet			
...	...	1,408	...	...	...	2,021	Ky. G. S. *Bull.* 1, p. 72.	Oil and gas in "Clinton" and Caney Sands. Salt water in Clinton. Probably stopped just short of Trenton.
...	...	...	...	...	...	2,597	Ky. G. S. *Bull.* 1, p. 78.	Gas in Pottsville Conglomerate and Berea sand.
...	...	...	...	...	...	2,770	W. Va. G. S. Vol. I (A), p. 495.	Oil and gas in Berea sand; gas in bastard limestone at 2,485 ft.
2,760	410	...	...	...	...	3,130	Unpublished.	Dry hole; oil show in Weir sand at 1,790 ft.; Corniferous and Niagara recorded together.
5,050	545	...	...	...	...	5,595	W. Va. G. S. Kanawha Report, p. xix.	Squaw sand gas; water in Niagara Lime at 5,592 ft.
...	...	...	...	...	...	4,105	W. Va. G. S. Raleigh Report.	Still drilling; base of Pottsville, 105 ft.
...	...	...	...	...	...	2,664	W. Va. G. S. Raleigh Report.	Dry hole.
...	...	...	...	...	...	2,000?	W. Va. G. S. Barbour-Upshur-Randolph Report.	Starts near base of Chemung Series. Shows of oil and gas.
...	...	...	...	...	...	2,385	W. Va. G. S. Barbour-Upshur-Randolph Report.	Starts near base of Chemung Series. Dry hole.
...	...	...	...	...	...	4,250	Not published.	Starts near base of Chemung Series. Pockets of gas at 825 to 835 ft., 1,053 and 3,830 ft. Strong salt water at 3,990 ft. to 4,000 ft.
...	...	...	...	...	...	4,570	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas in Big Lime, Big Injun, Berea, and Kane(?) sands.
...	...	...	...	...	...	4,386	W. Va. G. S. Barbour-Upshur-Randolph Report.	
...	...	...	...	...	...	4,028	W. Va. G. S. Doddridge-Harrison Report, p. 120.	Little gas in lime and shells at 2,950 ft.
...	...	...	...	...	...	3,955	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas show at 1,210 ft.
...	...	...	...	...	...	4,437	W. Va. G. S. Barbour-Upshur-Randolph Report.	Light gas in Big Injun sand; Kane(?) sand, 4,108 to 4,162 ft.
...	...	...	...	...	...	3,506	W. Va. G. S. Barbour-Upshur-Randolph Report.	Gas show, 2,425 ft.
...	...	...	...	...	...	4,450	W. Va. G. S. Doddridge-Harrison Report, p. 284.	Flow of oil in Speechley sand at 3,125 ft.
...	...	...	...	...	...	3,411	W. Va. G. S. Marshall-Wetzel-Tyler Report, p. 390.	Gordon sand oil; Speechley sand, 3,344 to 3,376 ft. with some dark oil.
...	...	...	...	...	...	4,500	W. Va. G. S. Vol. I, p. 364.	Base of Big Injun at 1,570 ft.; show of oil at 2,955 ft.
...	...	...	...	...	...	5,575	W. Va. G. S. Vol. I (A), p. 104.	Speechley sand, 3,150 to 3,165 ft.; Bradford sand, 3,500 to 3,520 ft. with trace of oil; hole stopped in Marcellus shale.
...	...	...	...	...	...	6,299+	G. S. A., *Bull.* Vol. 24, p. 275, 1913.	Gas, 4,850 ft., 4,870, 5,900, 5,905, 5,910, 5,915, 6,060 ft. Not yet completed.
...	...	...	...	...	...	3,880	W. Va. G. S. Vol. I	Slight shows of oil and gas.

depth of 1,000 ft. In 1890, when development was at its height, 3,000 ft. was considered an unusual depth, but by 1910 several holes had been drilled more than 5,000 ft. and at the present time at least two wells in the world have been drilled more than 7,000 ft. The advance in drilling has been just as rapid as in any other line of human endeavor where mechanical ingenuity and skill must be used and in these qualifications the Scotch-American well driller stands second to none. To say that drilling cannot be continued to greater depths than the Geary and Slaughter Creek wells would call for a total cessation of the enterprising experimentation that has characterized oil-well drilling from its very beginning.

Suggestions for Drilling

A comparison of the cross-sections on Plate 3, in conjunction with the data exhibited on Plates 1 and 2, shows that ideal conditions may be found along several of the anticlines east and west of the main Appalachian Basin, among which may be mentioned the New Martinsville, Washington, Brownsville, Arches Fork, Big Moses, Chestnut Ridge (known as the Warfield in southern West Virginia), and last and best of all, the Burning Springs, which stretches in a north and south direction for more than 30 miles through the very heart of the oil fields and in a region where the Devonian shales are probably much thinner than at points farther east. A well drilled at the point where this anticline crosses Walker Creek, near Sandhill, 10 miles southward from St. Marys, and 5 miles north of Petroleum on the Baltimore and Ohio Railroad, and 5½ miles south of the point where cross-section *CC′* crosses the anticline, would have a tidal elevation of 920 ft. Being at the summit of the great Volcano dome, it would start at an approximate interval of 1,000 ft. below the Pittsburgh Coal and should find the Big Lime of West Virginia at about 100 ft., the Berea sand at about 600 ft., the Corniferous limestone at about 3,500 ft., the Oriskany sandstone at about 3,550 ft., the Medina white sandstone ("Clinton") at about 4,700 ft., and should reach the Trenton limestone at about 6,000 ft. The dome-shaped structure at this point would almost insure the absence of water in the sands, at the same time making the financial risk much less than at any other point in the well-developed oil fields of the State. Oil or gas secured at this point would immediately open a prospective field for 30 miles along the crest of this great anticline where conditions are similar all the way from St. Marys on the Ohio River to Burning Springs near the corner of Wirt, Roane and Calhoun Counties.

Should this region prove to be commercially productive from the lower sands, attention would naturally turn to the other anticlines mentioned, where drilling would be more expensive but the possibility of finding oil and gas reasonably good.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Petrography of the Mount Morgan Mine, Queensland.

BY W. E. GABY, B. S., M. A.,* BUTTE, MONT.

(Arizona Meeting, September, 1916)

INTRODUCTION

SINCE the time of their discovery, the genesis of the ores at Mount Morgan, and the nature of the changes which have affected the surrounding rocks, have been the subject of investigation and speculation by geologists and mining engineers. Developments at this mine, for a long period the greatest gold producer of the world, show that with depth the orebody grades into copper, of which metal it now furnishes a large output, exceeding in value the gold.

In earlier times, the purity of the gold and the first extremely heterogeneous character of the ore made it an interesting problem. Now, the probable depth of the copper enrichment, genesis, and true relations of the neighboring rocks furnish equally profitable material for investigation, and the present somewhat detailed description of these ores and rocks, from microscopic study, may prove of interest.

The mine occupies a highly silicified portion of a belt of quartz-porphyry, which is believed to be the oldest rock of the region. This belt of quartz-porphyry and allied rocks runs north and south between two areas of a later intrusive granite, as shown on the geologic map,[1] Fig. 1. Near the orebody the quartz-porphyry has been intruded by a dense basaltic mass called the "Old Basic." Cutting all three of these igneous masses are two sets of basaltic dikes, one running northwest-southeast, and the other, cross dikes, northeast-southwest. These dikes may be contemporaneous with, instead of later than, the ore. Their number has caused peculiar difficulties in extracting it, as the clay selvage of decomposed material affords no adhesion between dike and ore, causing great blocks to loosen and come down.

The investigation was based on specimens furnished by B. Magnus, former general manager of the mine, and notes on the areal geology by

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* Assistant Geologist, Anaconda Copper Mining Co.

[1] Reproduced by permission of the Australasian Institute of Mining Engineers from the plate accompanying the paper of J. M. Newman and G. F. Campbell Brown, *Transactions, Australasian Institute of Mining Engineers*, vol. 15, part 2 (1910).

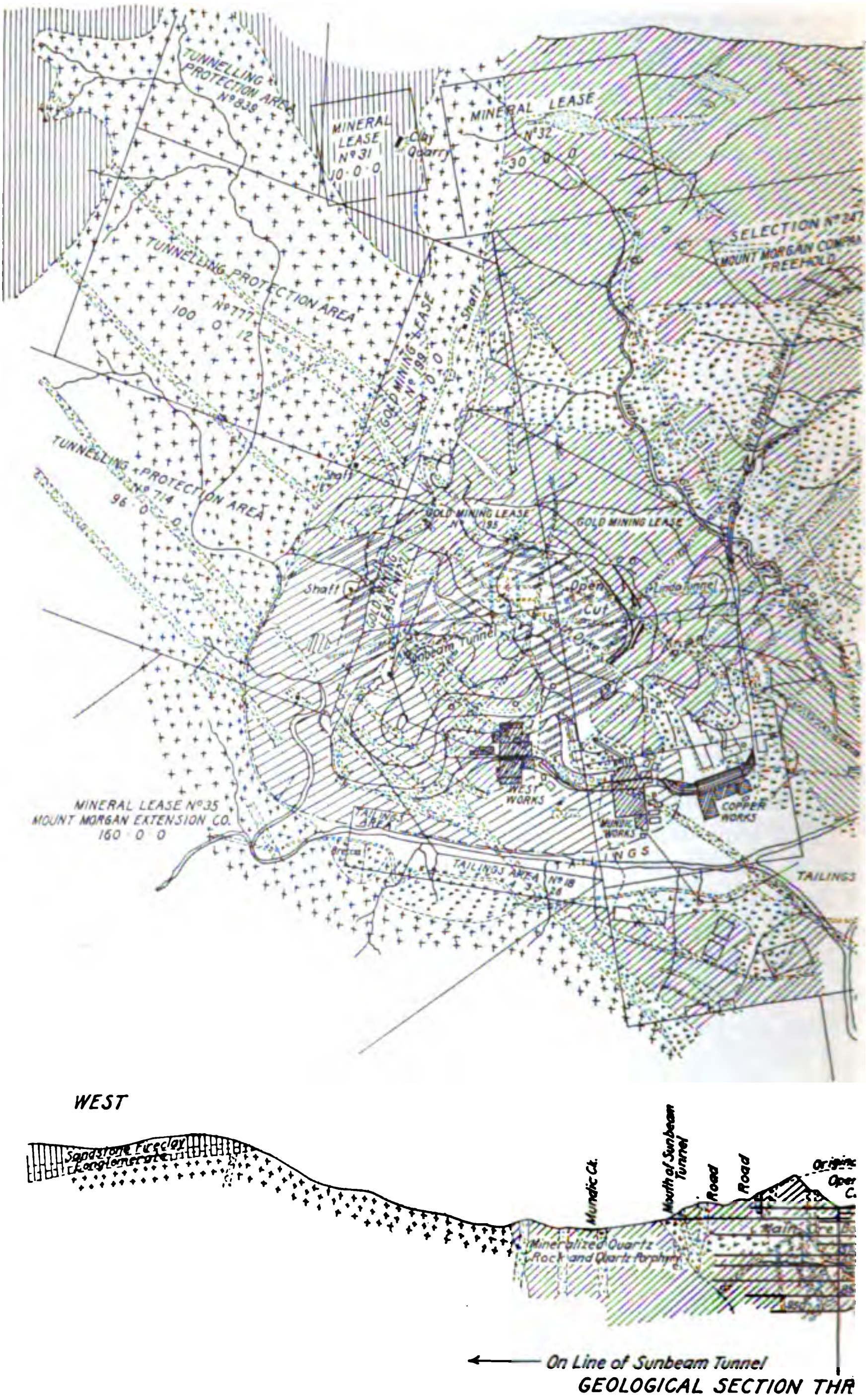

FIG. 1.—GEOLOGICAL MAP OF MOUNT MORGAN AND VICINITY.

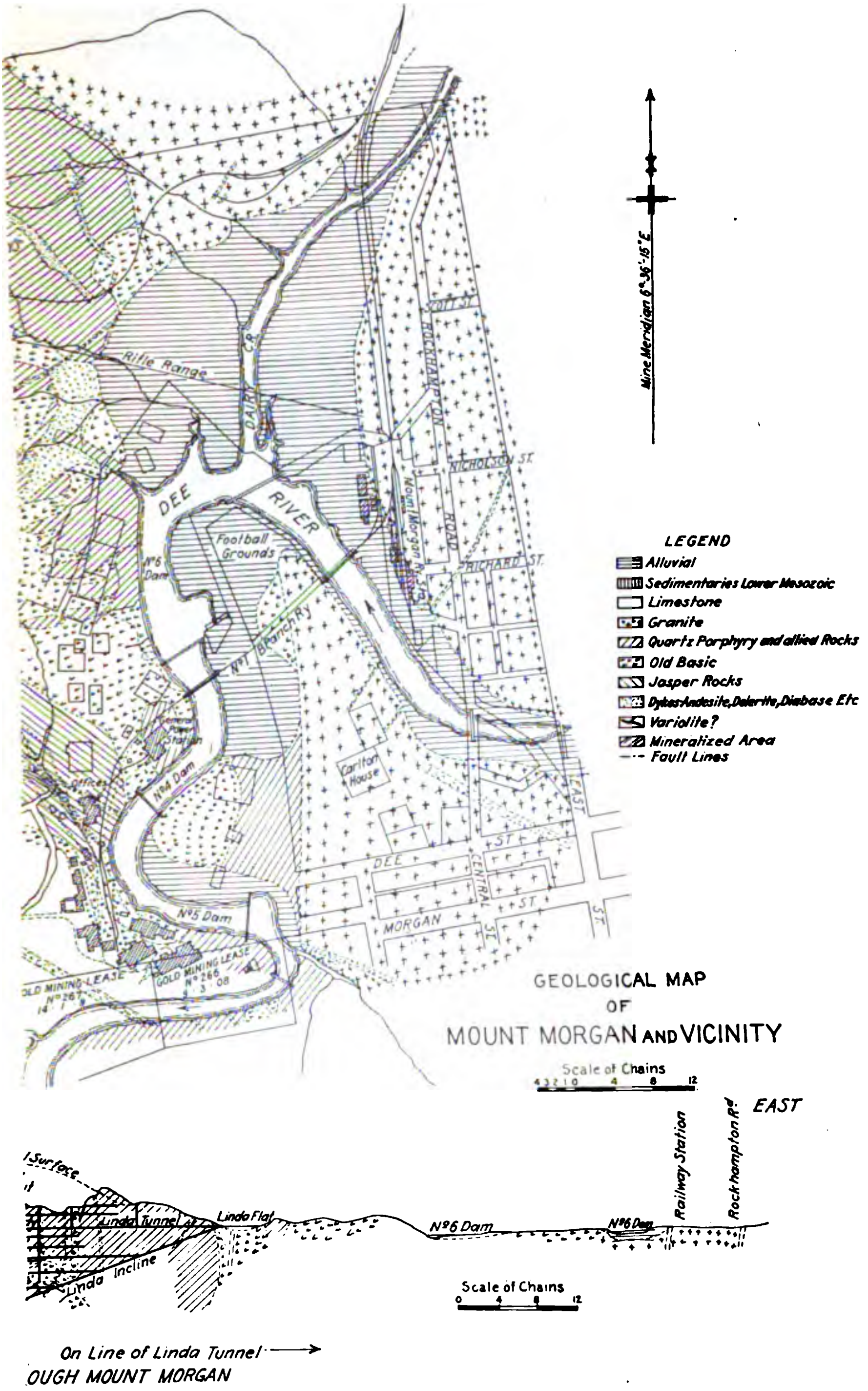

FIG. 1A.—GEOLOGICAL MAP OF MOUNT MORGAN AND VICINITY.

J. M. Newman and G. F. Campbell Brown.[2] Reference to the literature shows a wide divergence of opinion concerning its geology to have existed at different times.

SUMMARY OF OPINIONS ON GENESIS OF MOUNT MORGAN OREBODY

The first examination was made by R. L. Jack, as geologist for the Queensland Government, in 1884, and he came to a conclusion, from the evidence then available, that nothing but a thermal spring in the open air, or geyser, could account for the formation of the orebody. In 1891, Walter H. Weed of the United States Geological Survey was asked by Mr. Jack to examine and compare a suite of specimens from the mine with the siliceous sinters of the hot spring region of Yellowstone Park. Weed came to a similar conclusion, citing the Steamboat Springs of Nevada, which have long been known to be surrounded by deposits of sinter in fissures of which ore deposition is taking place.

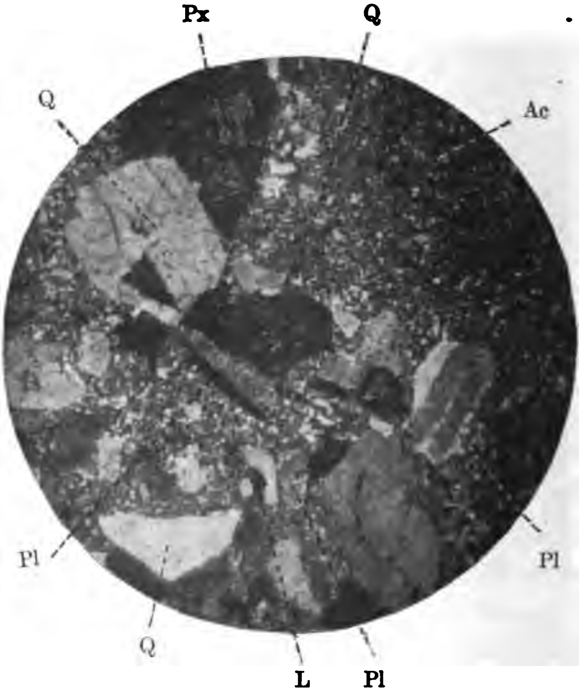

FIG. 2.—QUARTZ-PORPHYRY OF THE COMPOSITION OF QUARTZ-DIORITE OR QUARTZ-MONZONITE, WITH A GROUND-MASS OF FINE GRANULAR SECONDARY QUARTZ, CHLORITE, LEUCOXENE, AND HEMATITE. PLAGIOCLASE AND QUARTZ MAKE UP THE PHENOCRYSTS, AND PATCHES OF PYROXENE ARE COMMON. THE LATTER HAS BEEN PARTLY ALTERED TO CHLORITE AND EPIDOTE. CROSSED NICOLS. × 24.

T. A. Rickard, after an examination of the mine the same year, summarized the various views held up to this time as follows: 1. That the deposit is one of a geyser (R. L. Jack, 1884). 2. That it is an auriferous zone traversed by a series of quartz veins of auriferous mundic (gold-bearing pyrite) (J. MacDonald Cameron, 1887). 3. That it is the decomposed cap of a large pyrite lode. (The view held by several local and other mining engineers at the time of his examination.)

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[2] *Loc. cit.*, pp. 439 to 470.

Mr. Rickard dismissed the geyser theory as an altogether local appearance, based as it was on a small open cut which showed a fan-shaped arrangement of ore, and his own explanation of Mount Morgan was, that it "represents an altered portion of shattered country rock, which, by reason of its crushed condition, was readily acted upon by mineral solutions, and that these solutions replaced the basic and feldspathic with acidic and quartzose material which was also gold-bearing."

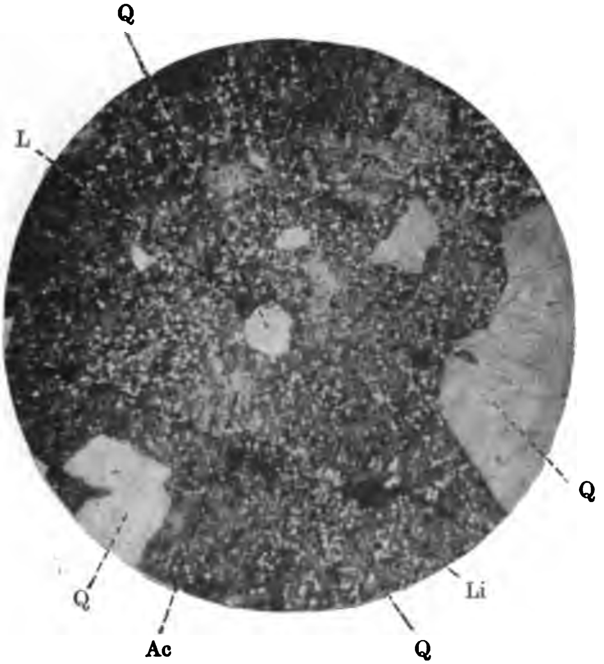

Fig. 3.—Quartz-Porphyry. There is a Strong Development of Actinolite among the Small Grains of Secondary Quartz Which Makes Up the Ground-Mass, Probably a Result of Proximity to the Basaltic Rocks. Leucoxene (White and Opaque, Hence Dark) is Abundant. Phenocrysts of Quartz, and Ground-Mass Largely Secondary Quartz. Limonite, as Evidence of Weathering, Occurs Disseminated and along Fractures. Crossed Nicols. × 24.

The geyser and replacement theories were later reviewed by Dr. Frank D. Adams of McGill University with an interesting account of the discovery and subsequent history of the mine. But by 1899, the reaching of depths below the zone of oxidation revealed data which did not agree with some of these earlier conceptions.

Many different names and origins have also been assigned to the rocks of the district. Metamorphism was supposed by some to have transformed original sedimentary rocks into the present ones, which are, as is generally stated, difficult to distinguish from igneous rocks. The supposed graywacke, quartzites, and shales of sedimentary origin are now known to be igneous or derived from igneous rocks. There are no true sediments near the mine except those to the northwest of it, which overlie the very latest of the intrusive masses. These sediments are Lower Mesozoic in age and consist of conglomerates, fireclays, and sandstones, with a possible bed of volcanic ash or tuff. No specimen of the latter material was available, so this point could not be determined.

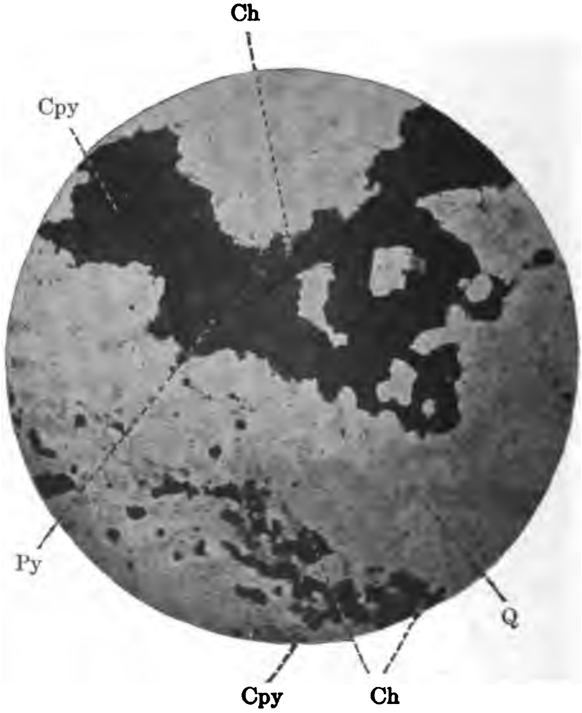

FIG. 4.—ORE, SHOWING PYRITE AND CHALCOPYRITE WITH CHLORITE IN GANGUE OF QUARTZ, THE CHLORITE WITHIN AND AROUND THE SULPHIDES, ALL THREE APPEARING TO OCCUPY THE SPACES AND CHANNELS BETWEEN THE QUARTZ GRAINS. SOME SMALL GREEN NEEDLE-LIKE CRYSTALS, NOT PLAINLY SEEN, ARE PROBABLY ACTINOLITE. PLANE LIGHT. × 24.

The notes by Newman and Brown, and by G. S. Hart, in the *Transactions* of the Australasian Institute of Mining Engineers, from whose maps and descriptions the field relations are taken, represent the most recent published work on the mine.

It is hoped the additional facts here set forth, from microscopic study, may be of assistance in solving the problem of the origin of its ores, as well as give additional information concerning the associated rocks.

In some cases the names of rocks have been changed to more nearly fit their mineralogy and textures as revealed in thin section, the names here given according with the American usage. Acknowledgments are due the authors of the above papers, to Mr. Magnus for specimens and notes, and to Dr. Charles P. Berkey of Columbia University for his kindly assistance in the laboratory.

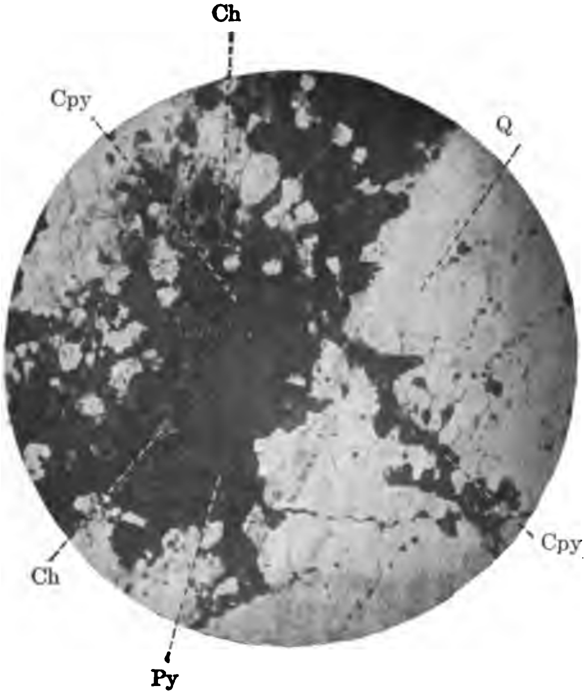

Fig. 5.—Ore, Showing Relation of Chlorite to the Sulphides, and Distribution of the Pyrite and Chalcopyrite with the Chlorite along the Veinlets and Spaces in the Quartz. Plane Light. × 24.

Description of Micrographs

The accompanying geologic map, Fig. 1, will make clear the general relationship of the Mount Morgan rocks, and with this in mind, the descriptions follow in the order of their relative geologic age.

Key to Minerals in Photomicrographs

Ac = Actinolite
Ap = Apatite
Aug = Augite
C = Carbonate
Ch = Chlorite
Cpy = Chalcopyrite
Hb = Hornblende
Il = Ilmenite
L = Leucoxene
Li = Limonite
M = Magnetite
Pl = Plagioclase
Px = Pyroxene
Py = Pyrite
Q = Quartz

Quartz-Porphyries and Ore

These rocks are grayish green and porphyritic with a fine-grained ground-mass and scattered irregular-shaped crystals of quartz. Two photomicrographs, Figs. 2 and 3, show them in thin section. Plagioclase is the predominant feldspar and ilmenite is the chief accessory mineral. Secondary development of quartz, actinolite, chlorite, epidote, kaolin, leucoxene, and hematite is a very pronounced characteristic. Later changes have developed some limonite, which is probably derived from pyrite, though occurring chiefly along fractures. Other fractures appear in the slides which are still more recent than these.

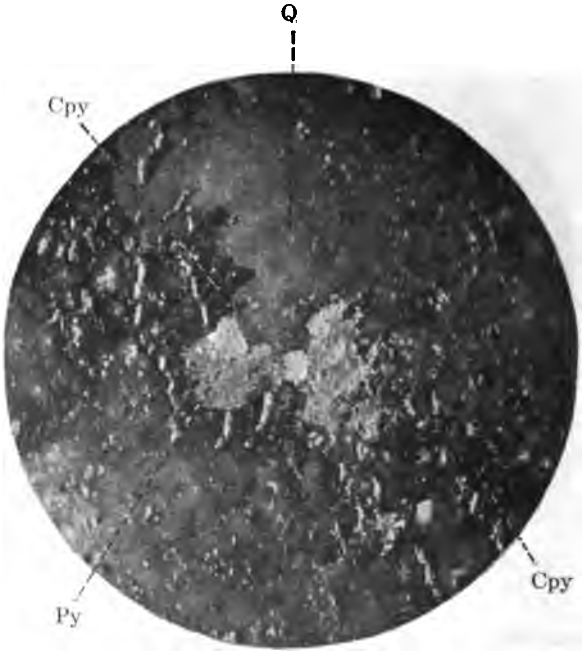

FIG. 6.—ORE. POLISHED SECTION SHOWING DISTRIBUTION OF PYRITE AND CHALCOPYRITE IN THE QUARTZ GANGUE. SILVER NITRATE HAS BEEN USED TO STAIN THE CHALCOPYRITE. THE MINERALS ARE DISSEMINATED AS REPLACEMENTS IN THE QUARTZ, AS WELL AS DISTRIBUTED ALONG THE VEINLETS AND CHANNEL BETWEEN THE QUARTZ GRAINS. BOTH SULPHIDES APPEAR TO HAVE BEEN DEPOSITED AT THE SAME TIME. REFLECTED LIGHT. × 24.

The belt of quartz-porphyry, which is the oldest of the rocks, is peculiar on account of the fact that the phenocrysts in it seem to have been subjected to a certain amount of primary fracturing. This would indicate that their original matrix had become somewhat viscous before movement within the mass entirely ceased, although all evidence of flow structure in the matrix or ground-mass has since been obliterated, if ever developed. Specimens of the unaltered rock would undoubtedly tell this. Nevertheless, its general heterogeneous nature suggests that it was originally a surface flow or large extrusive mass.

The quartz-porphyry within the boundaries of the ore deposit has been thoroughly silicified, and this gray quartzose material is streaked with veinlets and bands of pyrite and chalcopyrite. The original structure of the rock is only slightly preserved and Figs. 4 and 5 will show it to be replaced by aggregates of quartz grains of irregular size and shape.

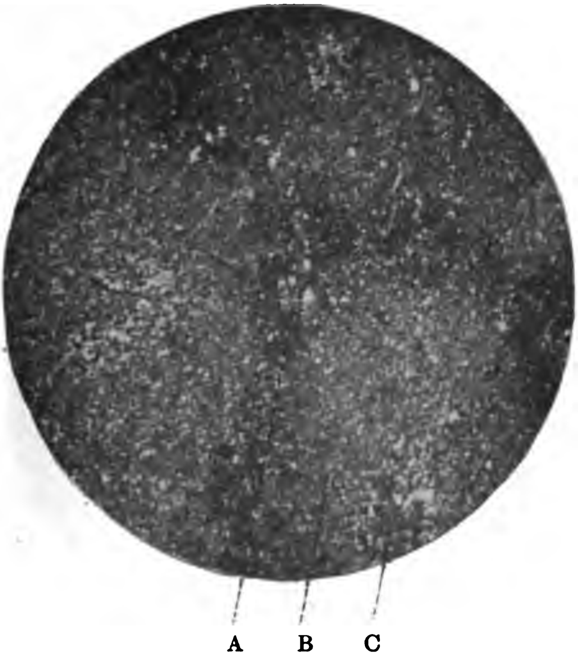

Fig. 7.—So-called Variolite, a Marginal Phase of the Quartz-Porphyry, Made Up of Secondary Quartz and Actinolite. The Apparent Peculiar Spherulitic Structure Shown in the Photograph is a Secondary Development. A, Actinolite in Fibrous Patches Bordering the Fine Quartz Aggregates. B, Fine Secondary Quartz. C, Coarser-grained Nucleus of Secondary Quartz. Crossed Nicols. × 24.

Veinlets occur which include chlorite and the sulphides. The banded nature of the secondary quartz is also apparent. The interior structure of the quartz grains, their outline and irregular shape, features which are not brought out clearly except under crossed nicols, suggest the replacement of porphyritic material.

The photograph of a polished section of the ore, Fig. 6, shows the distribution of the pyrite and chalcopyrite. The minerals occur partly disseminated as small crystalline aggregates in the quartz mass and partly concentrated along veinlets and channels between the quartz grains. They appear to have been deposited simultaneously, though somewhat later than the period of silicification of the quartz-porphyry.

The chlorite occurs within and around the sulphides as patches, and possibly actinolite along with it as needles, occupying with the sulphides the spaces and channels between the quartz grains. The chlorite is believed to have formed from the ferromagnesians of the original rock, incident with its silicification, and to have had some influence on the precipitation of the sulphides. As a mineral, it is too universal an alteration product to have a more direct significance. The actinolite will be discussed later in connection with the associated rocks.

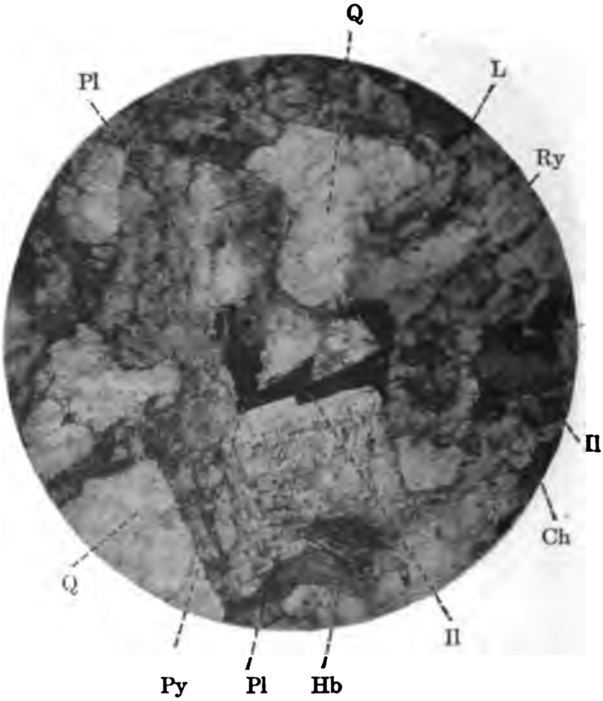

FIG. 8.—QUARTZ-DIORITE, TYPICALLY GRANITOID IN TEXTURE AND MADE UP OF INTERLOCKING GRAINS OF PLAGIOCLASE, QUARTZ, AND HORNBLENDE. ILMENITE AND APATITE ARE ACCESSORY MINERALS. PYRITE OCCURS IN THE ROCK AND MAY POSSIBLY BE PRIMARY, AS IT DOES NOT APPEAR TO CUT THE OTHER CRYSTALS. PLANE LIGHT. × 24.

A peculiar marginal phase of the quartz-porphyry has been developed at the northeast edge of the orebody near the dikes for which the name variolite, see Fig. 7, has been suggested on account of its peculiar spherulitic structure. But variolite is a variety of basaltic glass. The apparent spherulitic structure here is merely a peculiar secondary development of quartz and actinolite.

The Quartz-Diorite

The granite, on account of the predominance of plagioclase feldspar, is best termed a grano-diorite or quartz-diorite. It is a speckled, greenish-gray and white rock showing abundant hornblende and plagioclase, with quartz, in large interlocking grains. It shows some slight fracturing, and zonal growth of plagioclose. The feldspar is basic in composition, probably labradorite, and comparatively free from alteration. Some carbonization is observable in the feldspar, and the hornblende is largely changed to chlorite. Ilmenite and apatite are accessory minerals, some of the apatites being 2 to 3 mm. in length. Pyrite occurs in the rock and may possibly be primary, since it does not appear to cut structures in the other minerals to any considerable extent (see Figs. 8, 9, and 10).

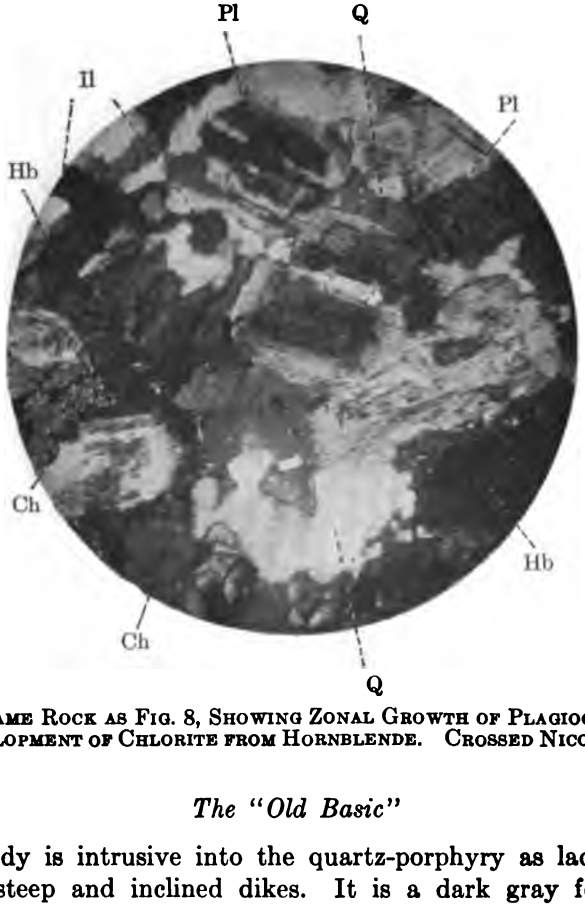

FIG. 9.—SAME ROCK AS FIG. 8, SHOWING ZONAL GROWTH OF PLAGIOCLASE, ALSO THE DEVELOPMENT OF CHLORITE FROM HORNBLENDE. CROSSED NICOLS. × 24.

The "Old Basic"

This body is intrusive into the quartz-porphyry as laccoliths and numerous steep and inclined dikes. It is a dark gray felsitic rock, weathering on its surface to a whitish limonite coating. Though felsitic, it does have a few very small feldspar phenocrysts (see Figs. 11 and 12). The rock is composed of small criss-crossed columnar crystals of plagioclase with some larger irregular grains of the same mineral. All are surrounded by pale green actinolite and chlorite, derived from altered pyroxene, and small grains of magnetite are scattered through the mass.

Quartz is present in small grains but is very largely of secondary origin. Small and sparse crystals of pyrite occur associated with patches of chloritic material and actinolite. The heavy iron content in magnetite attests to the basic character of the rock, the magnetite having also resulted from the alteration of the pyroxene. Evidently the ferromagnesian minerals were unstable under conditions which did not affect the feldspars. The rock is an unusual type, though composed of common minerals, and may be called an andesine basalt.

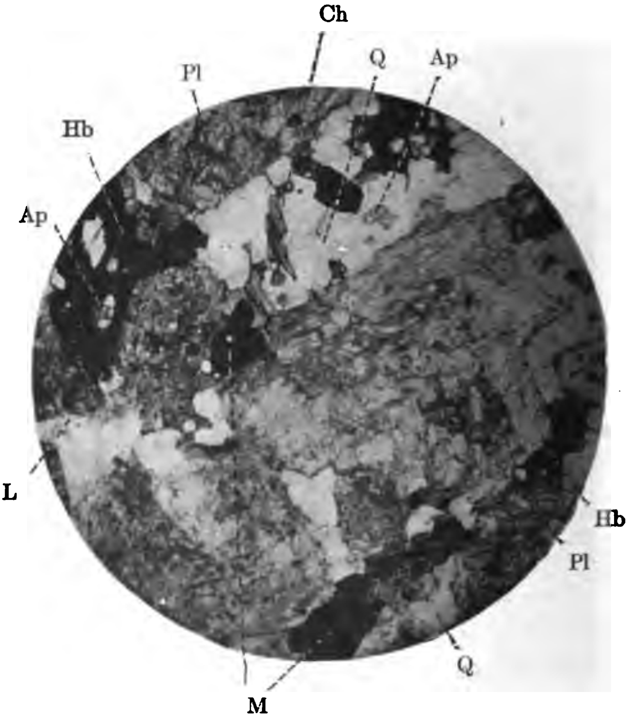

FIG. 10.—THIS PHOTOMICROGRAPH IS SIMILAR TO FIG. 8. IT SHOWS ABUNDANT PLAGIOCLASE, QUARTZ, AND HORNBLENDE. MAGNETITE, POSSIBLY ILMENITE, IS A PROMINENT ACCESSORY MINERAL, AND APATITE CRYSTALS OF FAIRLY GOOD SIZE ARE SCATTERED THROUGH THE ROCK. ALTERATION HAS AFFECTED MOST OF THE FELDSPARS TO SOME EXTENT, ATTACKING THEM ALONG STRUCTURE AND FRACTURE LINES. PLANE LIGHT. × 24.

Fig. 13 illustrates one of the thin sections under high power, and shows very clearly the small fibrous crystals of actinolite associated with the magnetite. As above stated, the actinolite, and probably most of the magnetite, is regarded as a metamorphic derivation from the original pyroxene in the rock, pseudomorphs of which are common through it. Pyrite also occurs in places penetrated by crystals of the actinolite, tending to the belief that this mineral, as well as the pyrite, is made up partly of introduced material. However, the pyrite may have formed around the actinolite at a later stage without replacing it.

Stope Dike

This rock, Fig. 14, belongs to the older dike series. It is dense, black and felsitic, very much resembling the "Old Basic," if indeed it is not the same rock. It includes an occasional fine patch of minute pyrite crystals, and weathers with a reddish iron stain. The structure of the rock may be described as a network of small columnar crystals of plagioclase and fibers of actinolite, filled in with irregular grains of quartz and magnetite. A few fine fractures appear which are healed with chlorite. Pyroxene was an original mineral, which, together with the feldspar, has been largely altered to quartz and actinolite.

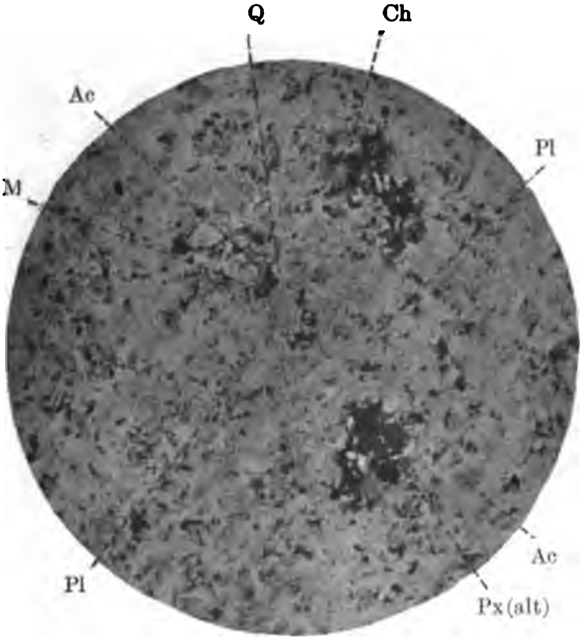

FIG. 11.—ANDESINE BASALT, SHOWING PYROXENE ALTERED TO SCATTERED AGGREGATES OF MAGNETITE WITH ACTINOLITE AND CHLORITE. FELDSPAR IS PROMINENT AND APPEARS TO HAVE BEEN THE FIRST MINERAL TO CRYSTALLIZE. ITS VARIETY IS ANDESINE. QUARTZ, LARGELY SECONDARY, IS PRESENT AS SMALL GRAINS THROUGHOUT THE MASS. THOUGH COMPOSED OF COMMON MINERALS, THIS IS AN UNUSUAL TYPE OF ROCK. PLANE LIGHT. × 24.

Dolerite Dikes

The material of these dikes varies slightly in composition and greatly in texture, some being diabasic and others basaltic. All the material is dark, massive and more or less felsitic. Fig. 15 shows the diabasic type. It is made up of rounded irregular grains of augite in a plexus of lath-shaped crystals of feldspar. The feldspar is plagioclase. The augite grains are small and irregular in outline, and appear to fill the spaces between the crossed crystals of plagioclase. Alteration has changed a considerable portion of it to chlorite and actinolite, part of which has replaced occasional lines and twinning bands in the plagioclase. Alteration has also changed some of the plagioclase to quartz. This quartz seems to have been derived from both pyroxene and feldspar, being penetrated by fibrous actinolite crystals of simultaneous growth. However, it can not be said that all the quartz is secondary, for in the case of many interstitial grains, the traces of a replaced mineral are too slight to make it possible to say it existed.

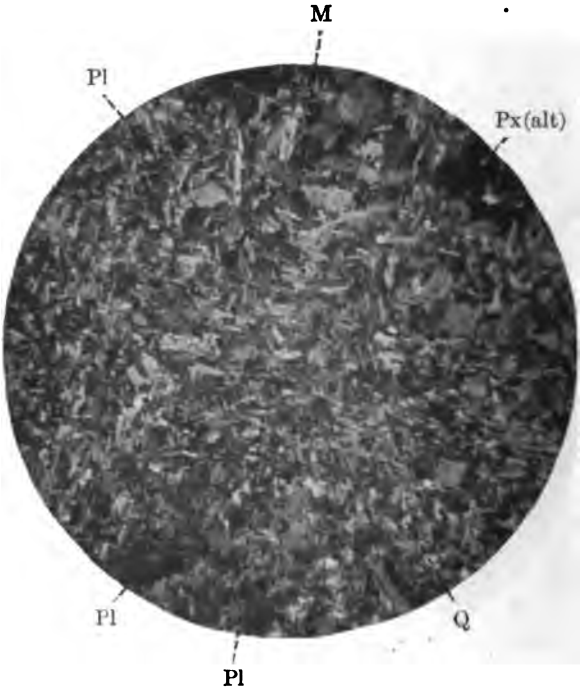

FIG. 12.—SAME ROCK AS IN FIG. 11. THE TEXTURE IS COARSELY FELSITIC TO PORPHYRITIC, AND THE CRISS-CROSSED ARRANGEMENT OF THE COLUMNAR PLAGIOCLASE CYSTALS IS APPARENT. THERE ARE A FEW PLAGIOCLASE PHENOCRYSTS VISIBLE, AND MAGNETITE IS DISTRIBUTED THROUGHOUT. CROSSED NICHOLS. × 24.

The term Dolerite, as used on the map, while indicating rock of this general character, does not sufficiently emphasize the important structural feature shown in the arrangement of the feldspars in this particular specimen, and it would, therefore, be more properly called Diabase. The idiomorphism of the feldspars with respect to the pyroxene, as shown by

Fig. 15, is also a distinguishing characteristic of diabase, along with their columnar structure and crisscross arrangement.

Similarly, for textural reasons, the dolerite shown in Fig. 16 is more properly called basalt. This material is a more felsitic variety of the rock composing this group of dikes. It seems to possess the same general alteration effects except that the quartz, if present, is not visible, which might argue that the quartz of the other basic rocks is entirely secondary. In the ground-mass are many fine rods of plagioclase, but as a whole it is finely felsitic and the rock a typical basalt. It contains some pyrite, which is generally taken to be an introduced substance.

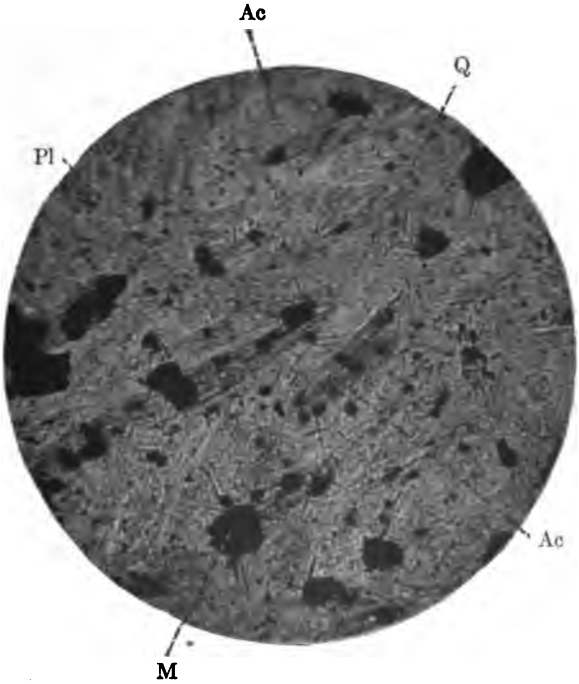

Fig. 13.—Same Rock as Figs. 11 and 12. The Higher Power Reveals Small Fibrous Crystals of Actinolite Associated with the Magnetite. The Quartz and Plagioclase are Rather Indistinct. The Actinolite, and Probably Most of the Magnetite, is Believed to be Derived from the Breaking Down of Original Pyroxene Under the Action of Siliceous Magmatic Waters. Plane Light. × 257.

The rock of the East Dike is a dark massive diabase, shown in Fig. 17. Quartz occurs in much the same relation to the crisscrossed columnar plagioclase as the pyroxene, and because this quartz may be partly primary the rock might well be called a quartz-diabase, which is a rather rare rock. In this original, the augite-pyroxene and plagioclase have been largely changed to chlorite, actinolite, and secondary quartz. The quartz and the unaltered plagioclase are penetrated to some extent by fibers of actinolite. The scattered magnetite is probably also secondary, and pyrite is present.

The variation in texture of these dike rocks is mainly due to different rates of cooling, occasioned either by the influence of wall rock, which may have been hot or cold, or by the thickness of the dikes themselves. Their chemical composition must also have exerted a strong influence on the order of crystallization of the minerals, producing the tendency to idiomorphic development of the feldspar.

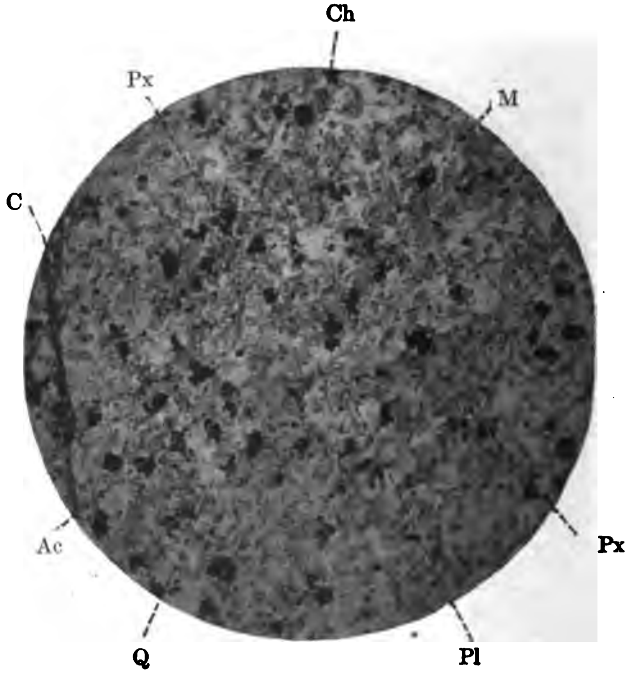

FIG. 14.—QUARTZ-BASALT, A FELSITIC COMPLEX OF PLAGIOCLASE AND PYROXENE WHICH HAS BEEN PARTLY ALTERED TO QUARTZ, ACTINOLITE, AND MAGNETITE, LEAVING ONLY A SMALL PART OF THE PLAGIOCLASE INTACT. PART OF THE QUARTZ, HOWEVER, APPEARS TO BE ORIGINAL. ABUNDANCE OF MAGNETITE, PART OF WHICH IS ALSO PRIMARY. SMALL CARBONATE VEIN. PLANE LIGHT. × 24.

SUMMARY

It will be seen from the descriptions above that, aside from the quartz-porphyry, the principal rocks associated directly with the ore are basaltic in character. They contain quartz, part of which I judge to be primary and part secondary. Magnetite is widely distributed and appears to have resulted in large part from the breaking down of the original ferromagnesian minerals. The rocks have undergone great change, but do not appear to have been much affected by superficial weathering.

The strong development of actinolite in all the specimens indicates either the alteration of pyroxene and plagioclase by solutions, or their change to a more stable mineral under conditions of regional metamorphism. The first is more probable, since there is not the slightest evidence of metamorphism in the surrounding grano-diorite. The pyrite in the basic rocks appears to be later than the actinolite, but as a whole is probably associated with the same period of alteration by hot solutions, the phenomena of expiring vulcanism. It is not known whether this pyrite contains gold or copper. In respect to the actinolite, it is not possible to say that the alteration is due to direct igneous influence; but because actinolite is generally regarded as a high-temperature mineral there is the possibility of regarding it in much the same light as tourmaline, on the presence of which in the volcanic copper deposit at Braden, Chile, the primary character of the deposit was regarded as certain.

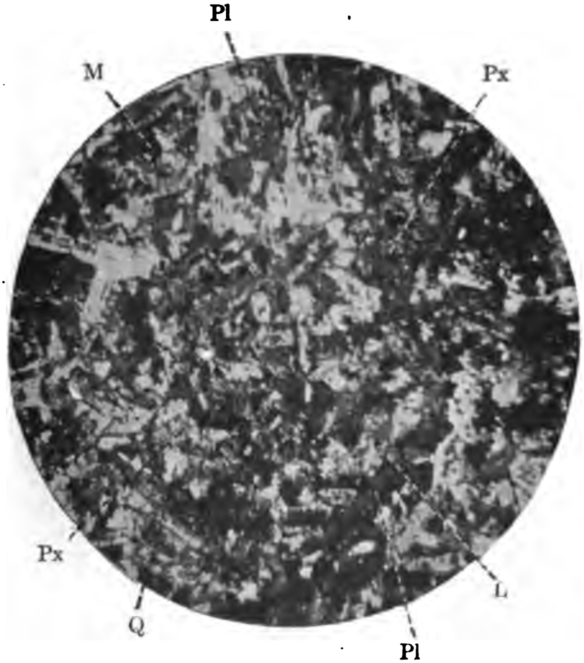

FIG. 15.—A DIABASE WITH THE CHARACTERISTIC TEXTURE. THE AUGITE GRAINS ARE SMALL AND IRREGULAR IN OUTLINE. MUCH OF THE PYROXENE HAS BEEN CHANGED TO CHLORITE AND SOME ACTINOLITE. TWINNING OF THE COLUMNAR PLAGIOCLASE CRYSTALS CAN BE SEEN. THE QUARTZ IS PROBABLY DERIVED FROM BOTH THE PYROXENE AND FELDSPAR, BEING PENETRATED BY FIBROUS CRYSTALS OF ACTINOLITE OF SIMULTANEOUS GROWTH. THE NAME DOLERITE IS AS GOOD A ONE AS ANY, BUT THE USAGE VARIES AMONG PETROGRAPHERS. CROSSED NICOLS. × 24.

The quartz in general seems also to be a result of metamorphic action on the plagioclase and pyroxene, penetrated as it is by the contemporaneous crystals of actinolite. Part of the quartz in the dikes, however, must be primary, as the sharp outline of some of the interstitial grains involve difficulty in conceiving the replaced original if these particular quartz grains were secondary. Hence the basic rocks might well be called quartz-basalts and quartz-diabases, which rocks are rather rare occurrences.

The grano-diorite or quartz-diorite intrusion shows some evidence of superficial weathering, but otherwise no alteration, and is considered a fairly fresh igneous rock. Pyrite is present in it, and the more altered character of the rock near the quartz-porphyries has led to the belief that its intrusion had something to do with the formation of the ores.

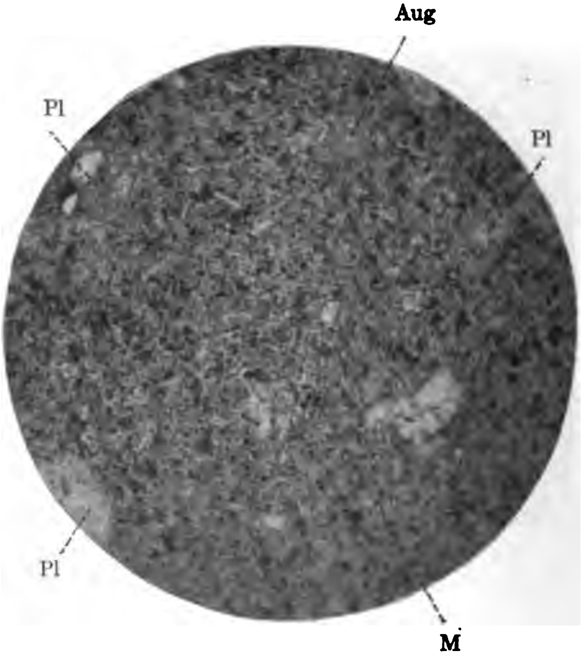

FIG. 16.—BASALT, A MORE FINELY FELSITIC VARIETY OF THE FOREGOING ROCKS, BUT CONTAINING SMALL SCATTERED PHENOCRYSTS OF PLAGIOCLASE. IN THE GROUND-MASS THERE ARE MANY LITTLE FINE RODS OF PLAGIOCLASE. QUARTZ, IF PRESENT, IS NOT VISIBLE. PLANE LIGHT. ×24.

The quartz-porphyries are thought to be such, rather than volcanic tuffs, because they contain no material foreign to the composition of a crystalline igneous rock, except the secondary quartz. By this quartz, however, these rocks are almost entirely replaced, the ground-mass completely so, and on this account their appearance under the microscope does somewhat resemble that of siliceous sinter. From the specimens examined and the mapping of this mass as quartz-porphyry, jasper, etc., its heterogeneous character, as of acid extrusive rock in general, is apparent. One of the specimens is a rhyolitic porphyry and another dacitic, and the structure of both, as already described, also suggests that the mass is an ancient surface flow of large extent. The later intrusions of quartz-diorite and the basic rocks have further altered its local character.

In regard to the sequence of the different igneous masses, it seems most reasonable from this study to put the "Old Basic" later than the grano-diorite and of the same age as the Stope Dike. No very strong structural evidence to the contrary is shown on the map, as the two apparent intrusions of grano-diorite into the "Old Basic" on the western and southern borders of the mineralized area may just as well be interpreted as intrusions of "Old Basic" around spurs of the grano-diorite, and the petrographic evidence we have in addition is very strongly in favor of placing the "Old Basic" in the same age relation to the grano-diorite as the two systems of basic dikes which cut it. All the basic-rock types show too marked a mineralogical and textural similarity, even in the character of their alteration, to be divided into two periods of intrusion by another rock of opposite character, the grano-diorite. However, such occurrences are not unknown in the history of rock magmas.

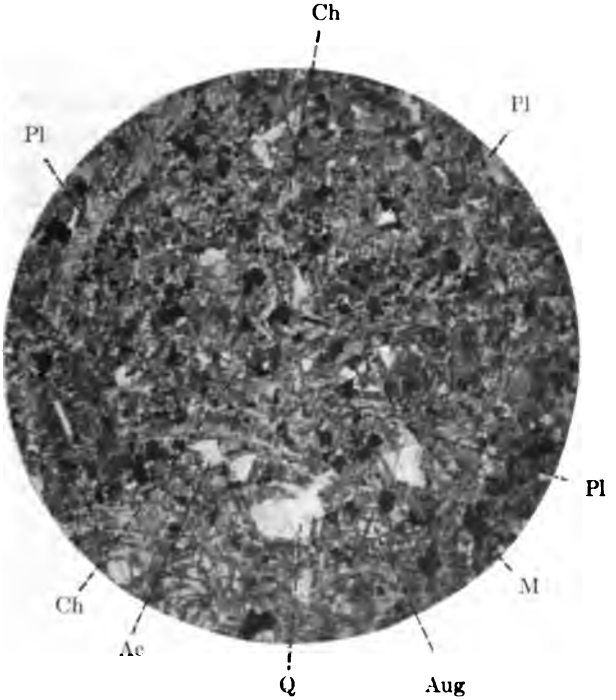

FIG. 17.—ORIGINAL QUARTZ-DIABASE IN WHICH THE AUGITE AND PLAGIOCLASE HAVE BEEN LARGELY CHANGED TO CHLORITE, ACTINOLITE, AND QUARTZ. THE DIABASIC TEXTURE IS EVIDENT, PYROXENE AND CHLORITE FORMING A FILLING BETWEEN THE CROSSED COLUMNAR CRYSTALS OF PLAGIOCLASE. ACTINOLITE NEEDLES PENETRATE THE QUARTZ WHICH, HOWEVER, IN SUCH A BASIC ROCK, MUST BE LARGELY SECONDARY. PLANE LIGHT. × 24.

According to this interpretation, the relative ages of the different masses are as follows:

Formation	Rock Name
9. Sedimentaries—Mesozoic.	
8. Faults.	
7. "East" Dike (Dolerite).	Quartz-diabase.
6. "North" and "South" Dikes (Diabase).	Quartz-basalt.
5. Andesite Dikes.	
4. "Old Basic," "Stope," "East and West" and "Cross" Dikes.	Andesine- and quartz-basalt.
3. "Flat" Dike.(?)	
2. Granite.	Quartz-diorite.
1. Porphyry.	Quartz-porphyry.

The ore is of granular secondary quartz containing pyrite and chalcopyrite with chlorite and actinolite, a highly silicified phase of the quartz-porphyry. No evidence of a brecciated or crushed zone appears in any of the slides studied. The enrichment has been derived from magmatic solutions, and there is no reason to believe that the pyrite and chalcopyrite were deposited at different times. The more abundant occurrence of the sulphides along spaces or channels between the grains of quartz, and in bands or streaks through the gangue, seems to indicate that they were deposited shortly after the quartz replacement, rather than at the same time. The associated chlorite and actinolite seem to have been derived mainly from the ferromagnesians of the original quartz-porphyry during the period of replacement by quartz.

The chlorite seen in the ore is, as stated before, believed to have no important significance, but referring to Figs. 2 and 3, the strong development of actinolite in the surrounding quartz-porphyry is believed to be closely associated with the sulphide mineralization. If such is the case, the presence of this mineral in the basic dikes might imply that they had to do with the enriching solutions rather than the earlier grano-diorite intrusion, as actinolite is a metamorphic mineral formed either as a result of hot water action or of direct igneous metamorphism. It is not uncommon to find it at the contact of basic with quartzose rocks, as, for instance, near the periodotite dikes in the Franciscan sandstone of California, which formation itself is highly metamorphosed.

It might be noted here also that many gold deposits are associated with basic rocks, and the further structural fact can not escape attention that the ore deposit here occurs near the intersection of two systems of basic dikes. Hence, to say the mineralization was contemporaneous with these dikes seems a closer correlation of the data than to associate it with the earlier grano-diorite, for there are numberless other points in the quartz-porphyry near the grano-diorite contact where enrichment might have occurred and did not do so.

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Shooting Soot from Stacks

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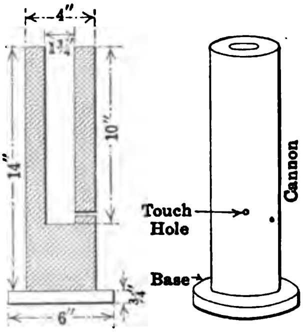

"The gun can be made out of an old piece of shafting about 4 in. in diameter and 14 to 16 in. long. Bore a hole 1¾ in. in diameter and 10 in. long in the center of the piece. Then bore a small horizontal hole ¼ in. in diameter through the piece to the bottom of the center bore. This hole serves as a touch hole for inserting the fuse.

"The whole thing can be mounted on a pedestal about 6 in. in diameter so that it will stand in an upright position.

"The method of operation is as follows: Pour some FFF Blasting Powder into the mouth of the cannon to about 2 in. from the top. Tamp to the collar with dry clay. A short piece of fuse is inserted in the touch hole and in contact with the main powder charge. Open the flue door at the bottom of the stack, set the cannon on the bottom and in the center of the stack, light the fuse and close the flue door.

"The explosion shakes and loosens the soot adhering to the sides, causing it to fall to the bottom. It can then be removed through the flue opening.

"A charge of 8 in. of FFF Powder, 1¾ in. in diameter, is sufficient for a stack up to 100 ft. high, and 4 ft. in diameter, or over. The number of shots necessary to clean a stack thoroughly depends upon its condition. Ordinarily three or four shots will clean a stack, but if very dirty it may require more. The size of the charge and length of the cannon can be regulated to suit the height and diameter of the stack."

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The capacity for overloads has been proven by tests showing that 200 per cent. load can be carried with only 10 per cent. increase in steam consumption.

Uniflow engines have been installed for rolling mills in Germany, one of the largest engines on record having a maximum continuous capacity of 6,300 hp. and a short-duration peak load capacity of 8,000 hp. The average load is 4,000 hp. This engine is installed at the Röchlingschen Steel & Iron Works at Völkingen, built by Erhardt & Sehmer.

A second engine of the same make was installed at the Rombacher Iron & Steel Works.

In this country the first uniflow engines for rolling mill work are those now under construction by the Nordberg Manufacturing Co., Milwaukee, Wis., for the Youngstown Sheet & Tube Co., Youngstown, O., the larger to drive a 12-in. rod mill, the smaller a 9-in. rod mill. The cylinder is a plain cylindrical casting, all valve chambers and steam passages being contained in the heads, bolted to the ends of the cylinder. The steam valves are of the Poppet type, the valve cages being separate castings ground into the heads with a steam-tight joint. The valves are of the double-beat, balanced Poppet type and no packing is required for the valve stems, which are accurately ground and polished and work in ground and lapped bushings. The valves are operated from lay shaft by a releasing valve gear with spring dashpots, the cut-off being under control of the governor through the wide range of loads. The steam enters the cylinder at the bottom, sweeping up over the heads and jacketing them before entering the valve at the top. The exhaust is through the ports in the center of the cylinder uncovered by the piston near the end of its stroke. Relief valves of the Poppet type with cataract dampening device are located near the bottom of the heads. In case of loss of vacuum and over-compression, these valves automatically open and connect the clearance space of the cylinder with the steam space in the head and the communicating steam piping. The steam conditions will be approximately 170 lb. pressure, 75° superheat, and 20 in. vacuum, steam being condensed by barometric jet condensers.

The larger engine is 44 by 50 in., 110 rev. and will have a capacity from 700 hp. to between 3,000 and 4,000 hp. The weight, including 110,000 lb., 10-ft. flywheel, will be 480,000 lb.

The second engine is a 37 by 48 in. at 110 rev. and its weight, including a 90,000 lb. flywheel, will be 360,000 lb.

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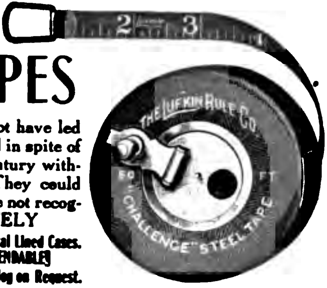

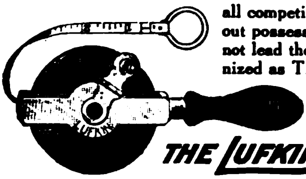

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Leschen's Hercules of June, 1916, contains the following interesting bit of information about the use of wire rope, in a short article about recapping rope that is attached to a cage in a hoist, or to cars on a slope, by the ordinary method of a socket fastened to the end of the rope by spelter.

The wires at this point ultimately become crystallized, as the vibration in the rope comes to a sudden stop at the socket, and the moisture that is likely to concentrate around the socket develops intensive corrosion.

The Leschen company advises, in the interests of safety, that at least once in 6 months a section of rope next to the socket be cut off and the socket re-attached to the cable, and they offer to send illustrated instructions to anyone sufficiently interested to make inquiries.

Trade Catalogues

The Ingersoll-Rand Co., has issued a catalog in Spanish under the title "Productos de la Ingersoll-Rand." This is attractively and substantially bound and contains 124 pages replete with illustrations, descriptive matter and tabulated data.

The catalog covers completely the company's line of air and gas compressors, vacuum pumps, reciprocating and centrifugal water pumps, rock drilling, metal and coal mining, prospecting and quarrying machinery, and pneumatic tools for machine, boiler shop and foundry work, and it ought to be of great service to Spanish-speaking users of pneumatic machinery.

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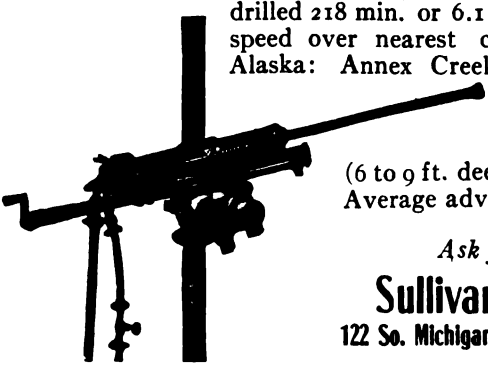

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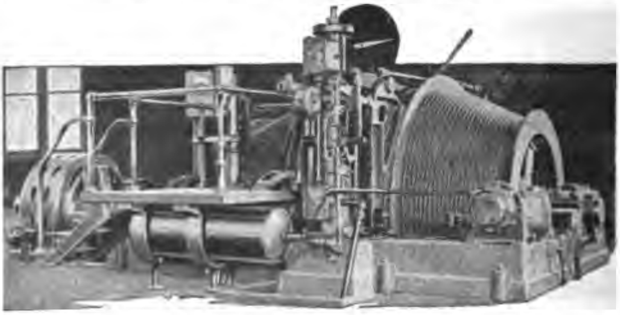

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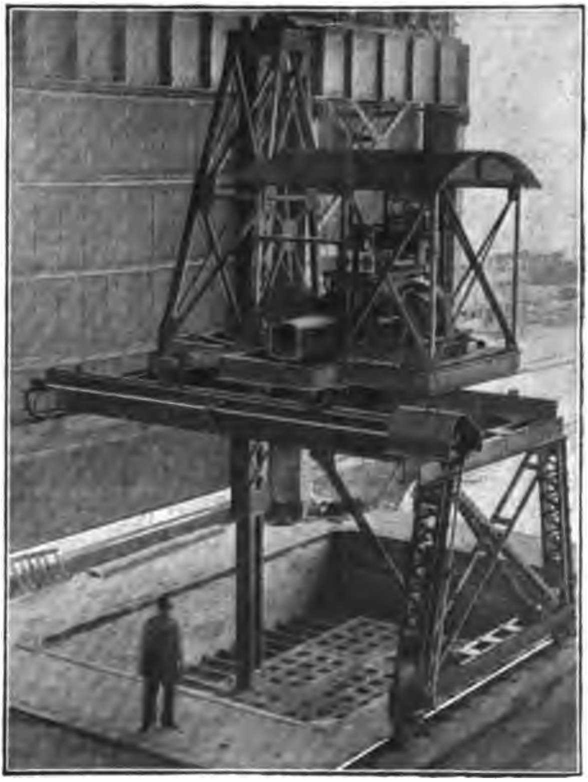

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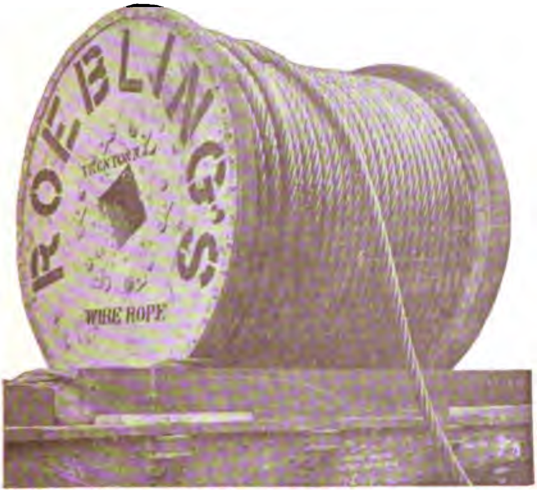

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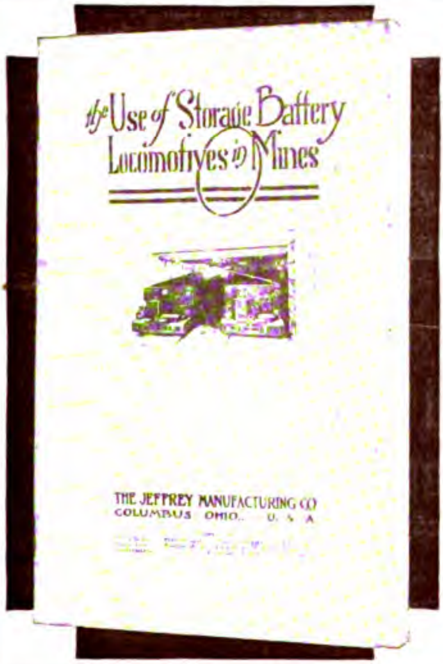
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OCTOBER, 1916

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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Discussion of the paper of Rudolf Gahl. By David Cole. 1870

Discussion of the paper of P. B. Liebermann. By Edwin M. Chance, Charles Legrand, and John Phillips 1875

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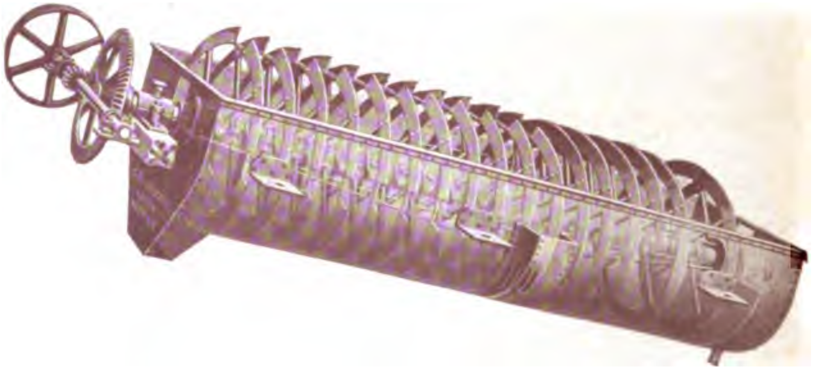

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SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

BULLETINS WANTED

The Institute is still in need of additional copies of the *Bulletin* for January, February and March, and will be glad to pay members the regular price of fifty cents per copy. Copies may be sent express collect, or postage may be used and the amount will be refunded in addition to the fifty cents.

TRANSACTIONS FOR 1916

Vols. 52 and 53 have been shipped to all who have paid their dues for 1916. Members who do not receive them within a reasonable time are asked to notify the Secretary.

MANUSCRIPT CLOSING DATE FOR THE NEW YORK MEETING, 1917

The 114th (New York) Meeting of the Institute will be held in the third week of February, 1917. The Committee on Papers and Publications has set Dec. 1, 1916, as the closing date for the receipt of manuscripts intended for this meeting.

Manuscripts must be received by the Secretary on or before this date. If they are sent through members of the technical committees they must be sent to those committees enough in advance of the time to be forwarded and in the hands of the Secretary by the time specified.

While Dec. 1, 1916, is the final closing date, it is a great advantage to get papers in as much before that time as possible as it enables them to be submitted in proof form to their authors. Papers received on or before

Nov. 1, 1916, will, as far as possible, be published in the January *Bulletin* or earlier, thus giving good distribution among the members in advance of the meeting.

The Institute's records show that the discussion of papers is very much better when they are distributed somewhat in advance of the meeting instead of in the *Bulletin* which goes out immediately preceding the meeting.

AMERICAN MINING CONGRESS

The American Mining Congress, of which the President is Carl Scholz, and the Secretary is J. F. Callbreath, and of which many of the officers and directors are prominent in Institute affairs, has extended to the members of the Institute a cordial invitation to attend the nineteenth Annual Session of the Congress, to be held at the La Salle Hotel, Chicago, Nov. 13 to 16, 1916, and to take part in its deliberations.

The morning sessions will be general meetings at which the attendance of all members of the Convention is hoped for, and which will be devoted to a discussion of subjects related to the three several heads, "safety," "efficiency" and "conservation." The afternoon meetings will be divided into several sections, each devoted to the discussion of subjects of interest to its special branch of the industry. It is planned to have all papers printed and distributed, allowing the author a limited time in which to present the salient features of his address and giving up the greater part of the time to a discussion of the subject presented by members on the floor. By this plan it is hoped to have a free expression of opinion and to provide ample time for reaching conclusions to be embodied in resolutions representing the best thought of the mining industry.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period August 1, 1916 to September 10, 1916:

C. L. Cantley, New Glasgow, N. S.
D. A. Cowell, Columbus, Ohio.
Alexander Deussen, Houston, Texas.
Chas. E. Dutton, Goldfield, Nev.
H. S. Emlaw, Grand Haven, Mich.
J. M. Gillette, Swatsburg, N. Y.
E. Giroult, San Antonio, Texas.
Bertram Grant, Kingman, Arizona.
Ruger W. Hay, Bisbee, Arizona.
C. F. Hendricks, Cobalt, Ontario.
Stanley C. Herold, San Jose, Cal.
Lloyd Hoffman, Pottersville, N. J.
Charles Hunter, London, England.
G. Jansen, Benkoelen, Sumatra.
T. H. Jenks, Oatman, Arizona.
Warren Jenny, Chanaral, Chile.
H. M. La Follette, La Follette, Tenn.
Montrose L. Lee, Montevideo, Uruguay.
V. H. McNutt, Billings, Mont.
J. W. Maller, New York, N. Y.
Edwin W. Mills, Tul. Mi Chung, Chosen, Korea.
Francis R. Pyne, Elizabeth, N. J.
Frank A. Ray, Columbus, Ohio.
John A. Rice, El Paso, Texas.
W. E. Rice, Ringwood Manor, N. J.
W. H. Staver, Lynchburg, Va.
Ralph H. Sweetser, Easton, Pa.
Kirby Thomas, New York, N. Y.
Henry Traphagen, Union Hill.
M. M. Valerius, Tulsa, Okla.
Wilfrid B. Wainewright, London, Eng.

Walter A. Barrows, Jr., of Brainerd, Minn., has been elected president and general manager of the Thomas Iron Co., Easton, Pa., to succeed Ralph H. Sweetser, whose resignation of several months ago becomes effective Aug. 31. Mr. Barrows has had an extended experience in blast-furnace and iron-mining work. He was for several years general manager of the Shenango Furnace Co., and for 5 years past has been engaged in iron mining and exploration in Minnesota.

Edwin S. Berry has become associated with Pope Yeatman and will engage in consulting mining engineering with offices at 111 Broadway, New York, N. Y.

Dr. G. H. Cox and **D. H. Radcliff** have opened offices in Tulsa, Okla., as consulting geologists.

W. R. Hamilton has withdrawn from the management of the Montobello Oil Co. He has opened offices in the Hobart Building, San Francisco, Cal., and will devote his time to general engineering work in petroleum and metal mining.

George B. Holderer, recently manager of the Furlough Development Co., in Arizona, is now with the General Chemical Co., 25 Broad Street, New York.

Joseph Jensen, mineral inspector in the Field Service of the U. S. General Land Office, has been appointed Assistant Professor of Mining at the Carnegie Institute of Technology, Pittsburgh, Pa.

Robert Livermore has resigned as manager of the Kerr Lake Mine, Cobalt, and is succeeded by H. A. Kee of the Nipissing Mining Co., Cobalt.

James MacNaughton, general manager and vice-president of the Calumet & Hecla Mining Co., is to take up his residence in Boston on Dec. 1. He will continue to manage the company. **John Knox,** superintendent, will take care of a considerable part of the work at the mine. Mr. MacNaughton will be a frequent visitor to the copper country and there will be no change in the titles of the two men.

Howard C. Parmelee, formerly Western editor of *Metallurgical & Chemical Engineering* of New York, has accepted the presidency of the Colorado School of Mines.

H. W. Ross has been appointed assistant manager for Backus & Johnston Co., Casapulca, Peru.

Lyon Smith has resigned as metallurgist with the Snyder Electric Furnace Co. of Chicago, and has gone to Florence, Colo., to be Assistant Superintendent with the River Smelting and Refining Co.

Thomas A. Stroup has left the engineering staff of the Tennessee Copper Co., and is now engineering assistant with the Utah Copper Co., at Salt Lake City, Utah.

William Thomlinson, managing director of the Seaton Carew Iron Co. Ltd., West Hartlepool, England, and the Carlton Iron Co. Ltd., of Ferry Hill, and Lieutenant Colonel in the Durham Light Infantry, raised and trained for Kitchener's army the 19th Durham Light Infantry (Bantams) and the 22d Durham Light Infantry (Pioneers). He was at one time at South Pittsburgh, Tenn.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, Cornell M. E., graduate, aged 31, married. Will soon complete 2 years' engagement in the Russian oil fields. Eight years' practical experience in drilling for and producing oil. Can speak Russian fluently and some French and German. Open for engagement. Interview after August 15, in the United States. No. 299.

Member, graduate mining engineer. Married. Aged 38. For past 6 years, general superintendent of large silver mine in Mexico. Desires connection with responsible mining company, preferably in the United States. Has had 15 years' experience in the United States and Mexico, including mine engineering, sampling, examinations, milling and mine superintendence. No. 312.

Metallurgist, 4 years' experience in electric furnace operation on steel, ferro-alloys and smelting of ores, both practical and research, desires position as metallurgist or superintendent. No. 314.

Member, technical mining graduate. 5 years' experience in coal and metal mines and construction work. Married. Aged 27. Employed at present. Interview in Salt Lake City or Denver. No. 316.

Member, American, aged 39, married, desires position as superintendent or mine superintendent. 14 years' experience in opencut and underground work. 12 years' Spanish, Portugese, Cuban and Mexican labor. Fluent Spanish. Good training in first aid and safety first. Specialty of high explosives and steam-shovel work. Good costs. Go anywhere, but tropics preferred. References. Interview, New York or Philadelphia. No. 318.

Member, aged 29, unmarried, technical graduate. 7 years' varied experience in metal mining: 3 years' as draftsman, surveyor and assayer; 4 years' operating experience as foreman, superintendent, and manager. Good health and habits. References. At present employed, available on short notice. Can handle men efficiently and get results. Employment with reputable company desired, preferably as foreman or superintendent. No. 319.

Member. Graduate Columbia School of Mines. Single. 30 years old. Six years' practical experience, 5 years in Mexico and Central America. Fluent Spanish. Good organizer, able to handle men. Desires position as superintendent or assistant in metal mine in U. S. or foreign country. No. 320.

Young technical graduate who has had 1 year's post graduate work in geology, would like to get into a geological department or to work as a mining engineer in engineering or mining departments. Willing to go anywhere. No. 321.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS
AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

COAL MINERS' POCKETBOOK. Ed. 11. New York, 1916.

CONSTRUCTION AND OPERATION OF A SINGLE TUBE CRACKING FURNACE FOR MAKING GASOLINE. Technical Paper No. 161, U. S. Bureau of Mines, Washington, 1916.

THE NEW ANACONDA. July, 1916. New York, Eugene Mayer, Jr. & Co., 1916. (Gift of publisher).

ELEMENTS OF MINING. By George J. Young. McGraw-Hill Book Co., Inc., New York, 1916. Price $5. (Gift of publishers.)

[NOTE.—This exceedingly useful little book will be found very valuable as a textbook and reference book for those who are taking up the subject of mining. It is written in a way which will commend itself to teachers and to those beginning the subject. It covers the fundamental principles and from them elaborates on different details of mining practice; the relation of geology to mining, and other related topics. The scope of the book is shown by the following list of titles of chapters:

Preface, Introductory, Prospecting, Boring, Drilling for Blasting Purposes, Rock Breaking, Transportation and Hoisting, Mine Drainage, Ventilation and Illumination, Support of Mine Workings, Open-Pit Mining, Alluvial Mining, Development, Underground Methods, Mine Organization and Operation, Mine Accounting, Accidents and Miners' Diseases, Examination of Mineral Deposits, Index. B. S.]

Geology and Mineral Resources

AMERICAN FERTILIZER HANDBOOK, 1916. Philadelphia, Ware Bros. Company, 1916. (Gift of publishers.)

COALS OF LETCHER COUNTY. Ser. IV, Vol. 4, pt. 1, Kentucky Geological Survey, Frankfort, 1916.

GEOLOGY OF THE SHAWNEETOWN QUADRANGLE IN KENTUCKY. Kentucky Geological Survey, Frankfort, 1916.

OIL SHALES IN THE PORT CURTIS DISTRICT. Publication No. 249, Queensland Geological Survey.

PETROLEUM PROSPECTS ON BRUNY ISLAND. Tasmania, 1916. Report by Arthur Wade.

PETROLEUM WITHDRAWALS AND RESTORATIONS AFFECTING THE PUBLIC DOMAIN. Bull. No. 623, U. S. Geological Survey, Washington, 1916.

RAPPORT SUR LES GISEMENTS DE CUIVRE DES CANTONS DE L'EST DE LA PROVINCE DE QUÉBEC. By J. A. Bancroft. Quebec, 1916.

THE ADELONG GOLDFIELD. Mineral Resources, No. 21, New South Wales. Department of Mines, Sydney, 1916.

THE SOUTH HEMMSKIRK TIN FIELD, with maps. Bull. No. 21, Tasmania. Geological Survey, Tasmania, 1916.

WOLFRAM MINES OF MOUNT CARBINE. Publication No. 251, Queensland. Geological Survey, Brisbane, 1915.

DETAILED REPORT on Raleigh County, Summers West of New River, and the Coal Area of Mercer County, by C. E. Krebs, assisted by D. D. Teets, Jr., with a chapter on Kanawha Marine Fossils by Wm. Armstrong Price, issued under date of August 1, 1916, with topographic and geologic maps. Eastern Raleigh, Summers and Mercer lie within the great New River and Pocahontas smokeless coal districts, while western Raleigh holds immense deposits of Kanawha Splint and gas coals. Price of Report with case of maps, including soil report and map, delivery charges paid by the Survey, $2.50. West Virginia Geological Survey, P. O. Box 848, Morgantown, W. Va.

General

CHART SHOWING PRICES FOR NON-FERROUS METALS FOR 1898–MARCH, 1916. (Gift of Daily Iron Trade.)

THE ENGINEER IN WAR. By P. S. Bond. New York, 1916.

METHODS OF SAMPLING DELIVERED COAL. Bull. No. 116, U. S. Bureau of Mines. Washington, 1916.

PRACTICAL LUBRICATION. By Lieut. G. S. Bryan. (Reprinted from Jour. Amer. Soc. of Naval Engineers, Vol. 27, Nov. 1915). (Gift of The Texas Company.)

TABLE OF THE EIGHTY-FOUR RESEARCH PAPERS AND SCIENTIFIC ADDRESSES BY SIR ROBERT HADFIELD, 1888–1912. (Gift of Sir Robert Hadfield.)

THEORY AND CALCULATION OF ALTERNATING CURRENT PHENOMENA. Ed. 5. By C. P. Steinmetz. New York, 1916.

TRAINING OF OUR CAPTAINS OF INDUSTRY. By Sir Robert Hadfield. (Reprinted from Iron and Coal Trades Review, June 9, 1916.) (Gift of Sir Robert Hadfield.)

Trade Catalogs

CHICAGO PNEUMATIC TOOL CO., Chicago-New York. Bull. 34-N. Single compressors steam and power driven.

DENVER FIRE CLAY CO., Denver, Colo. Catalog D. Case Furnaces, 1916.

E. I. DU PONT DE NEMOURS & CO., Wilmington, Del. Du Pont Magazine. Aug., 1916.

PELTON WATER WHEEL CO., New York, N. Y. Bull. No. 9. Pelton-Doble Centrifugal Pumps.

ROBERTSON-CATARACT ELECTRIC CO., Buffalo, N. Y. Illustrated brochure of buildings of company. pp. 21.

SMITH, H. COLLIER, Detroit, Mich. Catalog No. 50. Quickwork machinery.

SPARTA IRON WORKS CO., Sparta, Wis. Sludge bucket. Aug., 1916.

TEXAS CO., New York, N. Y. Lubrication. v. 2, No. 3; v. 3, No. 1–10. Nov., 1914; Nov., 1915. Aug., 1916.

UNDER-FEED STOKER CO., Chicago, Ill. Publicity Magazine. Aug., 1916.

MEMBERSHIP

New Members

The following list comprises the names of those persons who became members during the period Aug. 1, 1916, to Sept. 10, 1916:

Allen, Louis M., Jr., Chief Mine Sampler, Old Dominion Min. & Smelt. Co., Globe, Ariz.
Atkinson, Chester, Min. Engr., Asst. Supt., Smelt. Dept., Cerro de Pasco Mining Co., La Fundicion, Peru, South America.
Baker, Clair Lincoln, Genl. Mgr., Mexican Dept., Amer. Smelt. & Ref. Co., 1108 Mills Bldg., El Paso, Tex.
Barrows, Walter A., 3d, Asst. Supt., Blast Furnaces, Thomas Iron Co., Hellertown, Pa.
Bartholomew, George Pomeroy, Asst. Cons. Engr., Guggenheim Bros., 120 Broadway, New York, N. Y.
Billow, Elmer E., Pres., National Supply Co., 416 West Grand Ave., Chicago, Ill.
Brown, Samuel Rollins, Mill Supt., The Plymouth Cons. Gold Mines, Ltd., Plymouth, Cal.
Buchanan, R. H., Chief Engr. of Mines, Delaware & Hudson Co., Scranton, Pa.
Burns, William................Min. Engr., Phelps, Dodge & Co., Morenci, Ariz.
Campbell, Arthur Roy, Genl. Supt., Zinc Dept., United States Smelt. Co., 413 Republic Bldg., Kansas City, Mo.
Carlisle, George Lister, Jr., Mining..............14 Wall St., New York, N. Y.
Chapman, Raymond M., Chemist and Met., Owner, Indiana Laboratories Co., Ruff Building, Hammond, Ind.
Cleveland, Curtis, Chemist, International Lead Ref. Co., East Chicago, Ind.
Cody, Benjamin H., Chief Chem., The Arizona Copper Co., Ltd., P. O. Box 1003, Clifton, Ariz.
Coe, Harrison Streeter, Min. Engr., The Dorr Company, 810 Cooper Bldg., Denver, Colo.
Cole, Carl H., Asst. Met., Smelter Dept., Calumet & Arizona Mining Co., Douglas, Ariz.
Conner, William S., Min. Engr., Braden Copper Co., Rancagua, Chile, South America.
Cooke, Cosby Crittenden, Supt. of Coal Mines, The Low Moor Iron Co. of Va., Kay Moor, W. Va.
Cordes, Frank........Prest., Hubbard Steel Foundry Co., East Chicago, Ind.
Crawford, William Elmer, Mill Supt., Compania Beneficiadora de Pachuca, S. A., Apt. No. 1, Pachuca, Hidalgo, Mexico.
Crerar, George....................Hammond Mining Company, Glacier, Wash.
Cullen, Joseph F., Research Chemist, United States Smelt. Co., Midvale, Utah.
Deussen, Alexander, Geol., Humble Associates—Oil Producers, Room 504, Stewart Bldg., Houston, Tex.
Dick, James E............................Mgr., Akron Mine, Whitepine, Colo.
Dickson, Robert Henry, Min. Engr., Calumet and Arizona Min. Co., Bisbee, Ariz.
Duncan, Harold S., Min. Engr., Old Dominion Copper Mining & Smelting Co., P. O. Box 815, Globe, Ariz.
Eaton, Arthur..................Cons. Geol., 1641 Euclid Ave., Berkeley, Cal.
Edgar, Selwyn C., Jr., Pres., Edgar Zinc Co., 910 Boatmen's Bank Bldg., St. Louis, Mo.
Edwards, Hugh Robert......Cons. Engr., 433 California St., San Francisco, Cal.
Followill, Dexter Bemis, Supt. of Sulphate Dept., St. Louis Smelt. & Ref. Co., Collinsville, Ill.
Fry, Carl Herbert, Supt., Yellow Aster Mining & Milling Co., Randsburg, Kern Co., Cal.
Grenfell, Donald S........Chemical Engr., Mineral Point Zinc Co., Depue, Ill.
Grugan, Justice, Min. Engr., Mgr., Northern Ore Co., Edwards, St. Lawrence Co., N. Y.
Guyer, Raymond, Mining and examination, Guyer Hot Springs, Ketchum, Idaho.
Hamm, Karl G. C.............Chemist, Arizona Copper Co., Ltd., Clifton, Ariz.

HAMPTON, FRANK HOWARD FRANKLIN, Engr., Braden Copper Co., Rancagua, Chile, South America.
HARTMANN, MARIANO R., Owner and Mgr., Challacollo Silver Lixiviation Plant, and Mines, Cerro Gordo, La Granga, Prov. de Tarapaca, via Iquique, Chile, South America.
HAY, RUGER W.........Min. Engr., Calumet & Arizona Min. Co., Warren, Aris.
HENDRICKS, W. HOMER, Genl. Sales Engr., New Jersey Zinc Co., 55 Wall St., New York, N. Y.
HYSLOP, JOHN M............Met., Hubbard Steel Foundry, East Chicago, Ind.
JOHNSTONE, JAMES O., Chemist, International Lead Refining Co., 151st & McCook Sts., East Chicago, Ind.
KIRBY, ALFRED GEORGE, Min. & Met. Engr., 121 Howard Park Ave., Toronto, Ont., Canada.
KIRK, MILTON D., Asst. to Prest., The Davis Coal & Coke Co., Cumberland, Md.
KOBBE, WILLIAM H., Petroleum Engr., Pierce Oil Corpn., 420 Olive St., St. Louis, Mo.
LEMKE, CARL A., Asst. Supt. in Charge of Concentrating Dept., U. S. Smelt. Co., Midvale, Utah.
LINDMUELLER, CHARLES, Supt., Goldschmidt Detinning Co., East Chicago, Ind.
LLEWELLYN, PAUL, Works Mgr., Interstate Iron & Steel Co., East Chicago, Ind.
LOVE, ROBERT G., District Min. Engr., Delaware, Lackawanna and Western R. R. Co., Scranton, Pa.
McRAE, AUSTIN LEE ....Director, Missouri School of Mines, Rolla, Mo.
MACMICHAEL, HUGH RENWICK, Chief Engr., Southern Dept., American Smelt. & Ref. Co., 1108 Mills Bldg., El Paso, Tex.
MARSHALL, ALEXANDER FULTON, Chief Engr., Red Jacket Cons. Coal & Coke Co., Red Jacket, W. Va.
MARTINEZ, FIDEL C.......Asst. Met. Engr., Chino Copper Co., Hurley, N. Mex.
MINOR, CYRUS EDWARD............Engr., Mining Associates, Ltd., Bonanza, Colo.
MORGAN, DAVID......Supt., United Verde Extension Mining Co., Jerome, Ariz.
MORSE, GILBERT LIVINGSTON, Asst. to Mine Captain, New Jersey Zinc Co., Franklin, N. J.
MOTHERWELL, WILLIAM, Met......................... Colorado Springs, Colo.
OLMSTEAD, SEYMOUR G., .Chief Sampler, Utah Copper Co., Bingham Canyon, Utah.
OSBORN, WALTER X., Cons. Engr., United Verde Extension Min. Co., Douglas, Ariz.
PORTER, RUDOLPH.....................Mgr., Federal Lead Co., Federal, Ill.
PROCTOR, ISRAEL O..................Min. Engr., 1450 Dewey Ave., Butte, Mont.
RIDER, EZRA B., Min. Engr., Copper Queen Cons. Mining Co., P. O. Box 1224, Bisbee, Ariz.
ROBERTS, T. C., Chief Engr...........United Verde Copper Co., Jerome, Ariz.
ROBERTSON, JOHN H., Min. Engr......................Box 1405, Miami, Ariz.
ROSS, CLYDE POLHEMUS, Min. Geol., Asst., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
SANDFORD, JOHN J., Engr., Bunker Hill & Sullivan M. & C. Co., Kellogg, Idaho.
SAUNDERS, HOWARD P., Chief Engr. and Supt. Maintenance and Construction, St. Joseph Lead Co., Herculaneum, Mo.
SEAMON, WILLIAM H., JR., Mine Engr., Jos. S. Qualey & Co., Care Harding & McKee, 404 Roberts-Banner Bldg., El Paso, Tex.
SHEFFIELD, EUGENE S., Min. Engr., Departmental Mgr., Care Standard Oil Co. of New York, Producing Dept., Peking, China.
SPENCER, FRANK NORTON, Asst. to Genl. Mgr., The New Jersey Zinc Co., 55 Wall St., New York, N. Y.
SPRAGUE, CHARLES S., Pres. and Genl. Mgr., Jumbo Extension Mining Co., Goldfield, Nev.
STEWART, LEIGHTON, Min. Engr., Care Hayden Stone & Co., 25 Broad St., New York, N. Y.
VARLEY, THOMAS............................ Federal Lead Co., Flat River, Mo.
WAHL, ROY LESSLIE, Chief Min. Engr...........Inland Steel Co., Crosby, Minn.
WALKER, HENRY Y., Mgr.....................Tacoma Smelt. Co., Tacoma, Wash.
WAUCHOPE, ALBERT, Mine Supt., (Gold) Sons of Gwalia, Ltd., Gwalia, Western Australia.
WEBSTER, ERNEST B., Engr., with Highway Commission, Regina, Sask., Canada.
WEMPLE, LELAND E., MET.......American Zinc Lead & Smelt. Co., Hillsboro, Ill.
WYSOR, DAVIDSON CHARLTON, Geol., General Chemical Co., 25 Broad St., New York, N. Y.
ZIESEMER, HARRY M., Min. Engr., Copper Queen Cons. Min. Co., Bisbee, Ariz.

Associate Member

Coombe, A. Palmer, Care The Standard Oil Co., East Ohio Gas Bldg., Cleveland, O.

Junior Members

Brennen, George K. .... Scottdale, Pa.
Harrod, Wayne A., Flotation Operator, Engels Copper Mining Co., Keddie, Cal.
Hiraoka, Michiya, Prof. of Min. Engrg. & Met., Osaka Higher Technical College, Osaka, Japan.
Holmes, George F., Student...... Michigan College of Mines, Houghton, Mich.
Komar-of-Skye, George, Min. Engr., Care American Institute of Mining Engineers, 29 W. 39th St., New York, N. Y.
Kupferstein, Joseph T. .... Care Producers Oil Co., Tulsa, Okla.
Lau, Kaan .... 201 Bryant Ave., Ithaca, N. Y.
McDonald, Carlton K., Laborer, American Smelt. & Ref. Co., Omaha Plant, Omaha, Nebr.
Ovens, James Mason, Student, Michigan College of Mines, Houghton, Mich.
Repetti, George W., Min. & Met. Engr., The Dorr Company, 812 Cooper Bldg., Denver, Colo.
Scholl, Louis A., Jr., Geol. and Min. Engr., Producers Oil Co., Mayo Bldg., Tulsa, Okla.
Spurny, Emil .... 802 Second Ave., Astoria, L. I.
Stickney, William Herzog, Miner, Tramway Mine, Anaconda Mining Co., Butte, Mont.
Wilson, Harry R., Chemist, Care Burma Mines, Ltd., Nam Tu, Northern Shan States, Burma.
Woo, Wai Kyi .... 120 East St., Houghton, Mich.
Yost, Harold W., Min. Engr., C. H. Fulton, Experimental Zinc Smelter, Boatmen's Bank Bldg., St. Louis, Mo.

Total Membership, Sept. 10, 1916 5,748

CANDIDATES FOR MEMBERSHIP

Application for Membership.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

MEMBERS

The following persons have been proposed during the period Aug. 1, 1916, to Sept. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Arthur Abraham Arluck, Cleveland, O.
Proposed by J. Burns Read, F. A. Fahrenwald, Frank R. Van Horn.

Born 1886, Russia. 1907, Cleveland College Preparatory School. 1909, Spencerian Business College. 1913, Case School of Applied Science. 1913–14, Miner, Anaconda Copper Min. Co., North Butte Min. Co., Butte, Mont. 1914–15, Leaser,

Miner, Easton Pacific Mine, Virginia City, Mont. 1915–16, Const. Engr. and Surface Foreman, Cons. Interstate-Callahan Min. Co., Wallace, Idaho.
Present position: City Water Works Tunnel, Cleveland, O.

Theodore Newton Barnsdall, 2d, Bradford, Pa.
Proposed by W. L. Bell, Frank A. Ross, L. K. Armstrong.
Born, 1878, Titusville, Pa. Public Schools in Titusville, Pa., and Bradford, Pa. 1899–1912, Director, Manufacturers Gas Co., Bradford, Pa. 1906–08, Vice-Pres., Ardizzone Construction Co. actively engaged laying gas lines in Kansas. 1908–12, Pres., Barnsdall Construction Co. laying gas lines in Ohio, Kansas and Okla. 1910 to date, Treas., Highland Gas Co., Bradford, Pa. 1909 to date, Director, Potter Gas Co., Pittsburgh, Pa. 1910 to date, Oil Producer in Pennsylvania and Oklahoma.
Present position: Treas. and Genl. Mgr., Highland Gas Co.

Carl H. Beal, Muskogee, Okla.
Proposed by W. R. Hamilton, W. A. Williams, J. C. Branner.
Born 1889, Western Kansas. 1905–09, High School, Ukiah, Cal., and Palo Alto, Cal., 1909–12, Leland Stanford Jr. University, A. B. in Geology. 1914–15, Leland Stanford Jr. University, A. M. Geology. 1913, Asst. Geol., Kern Trading & Oil Co., Coalinga. 1913–14, Petroleum Geol., with Harry R. Johnson, Los Angeles, Cal., making geology reports on oil properties in developed and prospective fields of California, Montana, British Columbia and Alberta. 1914–15, Asst. Geol. and Mining, Stanford University, and during summer of 1915 in charge field work of summer class. 1915–16, Oil and Gas Inspector, U. S. Bureau of Mines, Muskogee, Okla.
Present position: Petroleum Technologist, U. S. Bureau of Mines, Washington, D. C.

Paul Thomas Benson, Thane, Alaska.
Proposed by G. T. Jackson, Joseph Daniels, E. V. Daveler.
Born 1889, Pueblo, Colo. 1908–11, Colorado School of Mines. 1911–12, Univ. of Washington, B. S. in Min. Engrg. 1912–13, Assayer and Asst. Chemist, Tacoma Smelt. Co. 1913–14, Assayer, Alaska Gastineau Min. Co., Juneau, Alaska. 1915, Foreman of Roll Dept., Alaska Gastineau Min. Co., Thane, Alaska. 1915–16, Foreman of Concentrating Dept., Alaska Gastineau Min. Co., Thane, Alaska.
Present position: Foreman of Concentrating Dept., Alaska Gastineau Min. Co.

Thomas Townsend Brewster, St. Louis, Mo.
Proposed by B. F. Bush, W. J. Jenkins, H. A. Wheeler.
Born, 1867, Saco, Me. Attended Common School. Experience and study. 1882–83, Blair & Truesdell, Bankers, Syracuse. 1883–86, Salt Springs National Bank, Syracuse. 1886–88, Acting Treas., Sweets Mfg. Co. (Rolling Mills), Syracuse. 1888–96, in business for myself, promoting, representing security holders in various corporations, specializing in coal propositions from 1890. 1896–98, Mgr., Thomas Carmichael, Trustee, Rio Grande Coal & Irrigation Co., Laredo, Texas. 1898–1912, Vice Pres. and Genl. Mgr., Rio Grande Coal Co. a reorganization of next above, property was sold in 1912. 1902 to date, Vice-Pres. and Genl. Mgr., Mt. Olive & Staunton Coal Co. 1910–12, Vice-Pres., Coal Operators' Assn. 5th and 9th Dist. of Illinois. 1912 to date; Pres., Coal Operators' Assn.
Present position: Vice-Pres. and Genl. Mgr., Mt. Olive & Staunton Coal Co.

Cecil John Brown, Wallaroo, South Australia.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1888, Moonta, South Australia. 1893–1901, State Public School. 1901–03, District High School. 1903–09, Technical, Mechanical Draftsman, Survey & Chemical Courses, Mechanical and Electrical Engineering. 1910–15, Civ. Engineering Diploma, International Correspondence Schools. 1903–05, Junior Draftsman's Dept., Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1905–08, Mining Engineering Draftsman (Wallaroo Mines) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1908–10, Surveyor, Surface & Underground (Wallaroo Mines) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1910–16, Engineering Draftsman & Surveyor (Wallaroo Smelt. Works) Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1907–16, Instructor, Art & Science Subjects, Moonta School of Mines, evening classes.
Present position: Chief Draftsman and Surveyor, Wallaroo Smelt. Works.

Godfrey L. Cabot, Boston, Mass.
Proposed by R. B. Woodworth, John Hays Hammond, I. C. White.
Born 1861, Boston, Mass. 1882, Harvard College, A. B. Magna Cum Laude in Chemistry. Studied in Zurich in 1883–84 with Dr. George Lunge and Prof. Victor

Meyer. Took graduate course in Harvard. 1882, Employed by Samuel Cabot, Jr. 1883, in experimental work in Natural Gas. 1886, in partnership with L. Cabot. 1886–87, made gas regulators. From then to present time busied in selling gas, building carbon black factories and operating them. Owning, operating, drilling gas and oil wells, laying pipe lines distributing and selling gas on an increasing scale, and am now busied also in liquefying Natural Gas.

Present position: Owner of five carbon black plants. Controlling owner of two others. Pres., Liquid Fuel & Gas Co.

Harry Elmer Bullock, Hazard, Ky.

Proposed by O. G. Petersen, William H. Shearman, Thomas T. Read.

Born 1883, Packard, Ky. 1903, educated at Lincoln Memorial Univ., Cumberland Gap, Tenn. 1904, Brown Univ., Providence, R. I. Made specialty of chemistry and mining engineering. No degree. 1906, Asst. to Genl. Mgr., Old Sterling Iron & Min. Co., Antwerp, N. Y. 1907, Supt., Bennett Jellico Coal Co., Trosper, Ky. 1908, Pres. and Genl. Mgr., Trosper Coal Co., Trosper, Ky. 1912, Mgr., Poteau Coal and Mercantile Co., Poteau, Okla. Also Genl. Mgr., Fort Smith Poteau and Western R. R.

Present position: 1914 to date; Pres. and Genl. Mgr., Kentucky Jewel Coal Co. and Kentucky Block Coal Co.

Norman L. Calder, Matahambre, Pinar del Rio, Cuba.

Proposed by J. Mark Smith, J. H. Jowett, L. D. Albin.

Born 1887, Hobart, Tasmania. 1906, Univ. of Tasmania. 1909–11, 1913–15, Columbia Univ., M. E. 1902–06, Tasmanian Smelt. Co., Tasmania. 1907–09, Transvaal Bischoff Tin Mine, Transvaal. 1911–12, Nevada Cons., McGill, Nev. 1912–13, Goldfield Cons., Goldfield, Nev. 1912–13, Ray Cons., Ray, Ariz. 1915, Minas de Matahambre, Matahambre, Cuba.

Present position: Asst. Engr., Minas de Matahambre.

Everett Carpenter, Bartlesville, Okla.

Proposed by Alfred J. Diescher, J. C. McDowell, Henry L. Doherty.

Born 1884, Holton, Kans. 1911, University of Oklahoma, A. B. 1908 and 1909, Asst. Geol., Oklahoma Geol. Survey. 1910, Junior Geol., U. S. Geol. Survey. 1911–16, Chief Geol., Wichita Natural Gas Co. and allied concerns.

Present position: Chief Geol., Empire Gas & Fuel Co.

Cairy Clyde Conover, Springfield, Ill.

Proposed by G. H. Cox, D. H. Radcliffe, M. M. Valerius.

Born 1890, Emporia, Kans. 1896–1908, Grammar School and High School, Carrollton, Mo. 1908–12, Missouri School of Mines and Metallurgy, Rolla, Mo., B. S. 1912, Mill man in wet mill and magnetic mill, Vinegar Hill Zinc Co., Platteville, Wis. 1913, Chemist, Chief Chemist, and later Asst. Supt., National Zinc Co., Springfield, Ill.

Present position: Asst. Supt., National Zinc Co.

Ralph Augustus Conrads, Garfield, Utah.

Proposed by R. C. Gemmell, H. C. Goodrich, R. H. Hawley.

Born 1880, Trenton, Mo. 1897, Grad., High School, Trenton, Mo. 1904, Grad., Missouri School of Mines, B. S. in Min. Engrg. 1902, Rodman, on R. R. Construction for Arkansas & Choctaw Ry. in Indian Territory. 1904–05, Concentration and Assaying, Utah Copper Co., Bingham Canyon, Utah. 1905–06, Engr., Balaklala Cons. Copper Co., Shasta Co., Cal. 1906–07, Engrg. and Cyanide mill, The Annie Laurie Gold Min. Co., Kimberly, Utah. 1907, Engr., Cia. Carbonifera Agujita, of South America, Sabinas, Coah., Mex. 1907–11, Engr., for 8 months, remainder of period, Asst. Mgr., Esperanza Min. Co., El Oro, Mex. 1911–12, Mining in interest of myself and associates in Ter. de Tepic and Oaxaca, Mex. 1912–13, Asst. Mgr., Cia. Minera Las Dos Estrellas, El Oro, Mex. 1913–14, Mgr., Cia. Minera Santa Ana Esperanza, Zac., Mex.

Present position: Met. Engr., Magna Plant, Utah Copper Co.

John Vipond Davies, New York, N. Y.

Proposed by W. L. Saunders, Alexander C. Humphreys, B. F. Cresson, Jr.

Born 1862, Swansea, South Wales. Wesleyan College, Taunton, England. Matriculated—University of London. 1880–84, Articled Apprentice, Parfitt & Jenkins of Cardiff, South Wales, Contracting Engineers. Principal work, Junction Dry Dock-Dock Gates, Bule Docks, Cardiff—Shaft 2,500 ft. deep, 25 ft. dia. Ynysybwl

Colliery and machine shop practice. 1884–85, Mine Surveyor, Collieries, John Vipond & Co. 1885–88, Asst. Engr. & Chief Asst. Blaenavon Coal & Iron Co., South Wales, in charge erection and construction blast furnaces, stoves, coke ovens, regen. furnaces, rolling and tire mills—Bessemer convertors and steel plants—Limestone quarry operations and brick making plants. 1888–89, Sea-going Engr., S/S "Argus" of Melbourne. 1889–90, Engr. & Mgr., Anthracite Coal Briquetting plant, Mahanoy City, Pa. 1891 to date; Charles M. Jacobs as Cons. Engr. Principal works: East River Gas Tunnel from 70th St., New York to Ravenswood; Chief Engr., designed and built, W. Va. Short Line R. R.; Chief Engr., designed and built, Kanawha & Pocahontas R. R.; Chief Engr., designed and built, Atlantic Ave., Improvement, L. I. R. R.; Chief Engr., designed and built, Entire tunnel and subway system of Hudson & Manhattan R. R., New York and New Jersey. 1906–07, Two years in charge for Contractors of Grand Central Yards Improvements. 1904, Cons. Engr., Board of Water Supply Detroit on Intake Tunnel. Engineers in charge of construction Hales Bar Dam across Tennessee River near Chattanooga involving pneumatic pressure construction for foundations. Engineers in charge of construction of 19 miles Aqueduct Tunnels for Mexico City Tramways Co. to divert waters Laxaxalpam River to Necaxa Dam. Engineer of design and actually in charge Construction Astoria Tunnel under East River, N. Y., for Astoria L. H. & Power Co., Tunnel about 20 ft. dia., 4,700 ft. long, Shaft 250 and 280 ft. deep. Consulting Engineer N. Y. Municipal Rw. Co. (B. R. T.) on Subway Railway developments in City of New York. Member Engineer Board Moffat Tunnel Commission, Colorado, reporting on Moffat Tunnel through Rocky Mountain Divide (other members, D. W. Brunton & Finch).

Present position: Vice-pres., Jacobs & Davies, Inc.

Graham Barclay Dennis, Spokane, Wash.

Proposed by L. K. Armstrong, Charles H. Goodsell, F. A. Ross.

Born 1856, Southampton, England. 1865–75, Cincinnati Public Schools. Cincinnati Pharmacy School, Bethany College, West Va. 1875–78, Asso. Editor, Business Mgr., Journal, Dayton, O. 1879–80, Editor and Publisher, Farmers' Home, Dayton, O. 1880–85, Investment Bankers, G. B. Dennis & Co., Dayton, O. 1885, Editor and Publisher, Spokane Miner, Spokane, Wash. 1886–88, Pres. and Mgr., Muscovita Mica Co., Spokane, Wash. 1889–91, Pres. and Mgr., Ross Park Elec. Ry. Co., Spokane, Wash. 1891–1916, Pres., Old Dominion Min. & Mill. Co., Spokane, Wash. 1893–96, Pres., Northwest Min. Asso., Spokane, Wash. 1899–1916, Pres. and Mgr., Insurgent Gold Min. Co., Spokane, Wash. 1908–16, Pres., Warehouse & Realty Co., Spokane, Wash. 1916, Pres., Spokane Mining Mens' Club, Spokane, Wash. 1916, Secy. & Treas., Spokane Railway & Terminal Co., Spokane, Wash. 1890, Vice-Pres., Northwestern Industrial Exp. Co., Spokane, Wash. 1906–16, Director, Exchange National Bank, Spokane, Wash. 1912–13, Vice-Pres. for Washington, American Mining Congress, Spokane, Wash. Have held like positions, within the years named, and many others of responsibility and trust. Trustee and Treas., Jenkins Univ., Spokane, Wash. Member, Com. Revision Federal Mining Laws. Organizer and Director and Pres., Spokane Publicity Bureau, for the years 1904–08. Some experience and Real Estate.

Present position: A number of above, and Pres., Spokane Mining Mens' Club.

Alfred F. Duggleby, Santiago de Cuba, Cuba.

Proposed by Max Roesler, S. B. Patterson, Jr., J. T. Singewald, Jr.

Born 1893, Brisbane, Aust. 1915, Colorado School of Mines, E. M. 1910–11, Draughtsman, American Pneumatic Action Co., Davenport, Ia. 1911, Transitman, A. R. Boudinot, Davenport, Ia. 1912, Miner, with E. P. Arthur, Jr., Cripple Creek, Colo. 1913, London Mines Co., Apex, Colo. 1914, Transitman, City Eng. Office, Davenport, Ia. 1915, Research in concentration Titanium Ores, Titanium Alloys Co. in connection with F. W. Traphagen, Golden, Colo. 1915–16, Mgr., Gold Reef Mines Co.

Present position: Mine Supt., Juragua Iron Co.

David L. Dunn, Joplin, Mo.

Proposed by H. A. Buehler, G. B. Corless, A. F. Truex.

Born 1881, Braidwood, Ill. 1897, Grad., High School, WeirCity, Kans. 1902–05, Asst. Engr., Central Coal & Coke Co., Weir City, Kans. 1905–06, Resident Engr., Central Coal & Coke Co., Bevier, Mo. 1907–09, Resident Engr., C. E. & C. Co., Huntington, Ark. 1910–11, Engr., and Asst. Supt., C. C. & C. Co., Rock Springs, Wyo. 1912, Asst. Chief Engr., C. C. & C. Co., Kansas City, Mo. 1913, Resident

Engr., C. C. & C. Co., Pittsburg, Kans. 1913–15, Supt., C. C. & C. Co. Mines, Pittsburg, Kans.
Present position: Stockholder and Manager, Spengeon Tiger M. Co. and Bumble Bee M. Co.

Harold Clinton Eddy, San Francisco, Cal.
Proposed by Arthur F. L. Bell, Albert Burch, M. E. Lombardi.
Born 1882, Middleboro, Mass. 1902–04, Colorado School of Mines. 1905, Mass. Inst. of Technology. 1907–08, Colorado School of Mines, E. M. 1909–10, Supt. Nevada Belle Helen Mines Co. 1910–11, Supt., Orleans Petroleum Co. 1911–13, Victor Electric Co.
Present position: 1913 to date; Engr., Petroleum Rectifying Co. of California.

Jay L. Emrich, Durango, Colo.
Proposed by F. C. Gilbert, L. R. Clapp, P. A. Mosman.
Born 1888, New York, N. Y. 1908–12, Colorado School of Mines, E. M. 1912, San Pedro Gold and Copper Min. Co., San Pedro, N. Mex. 1913, American Smelt. & Ref. Co., Leadville, Colo. 1913–16, American Smelt. & Ref. Co., Denver, Colo.
Present position: Chief Chemist, American Smelt. & Ref. Co.

Fred A. Fair, Boulder, Colo.
Proposed by M. S. Brandt, R. D. George, George W. Teal.
Born 1878, Ottumwa, Ia. 1898, Grad., East Denver High School. 1899–1902, School of Mines, Golden. 1905, classified minerals, as assistant, Univ. of Colo., Boulder. 1903–04, Asst. Engr., George Evans Gold Pan Placer, Breckenridge, Colo. 1905, U. S. Deputy Mineral Surveyor, Colo. 1906, Charge of construction city water system of the City of Boulder $500,000 plant. 1907–08, City Engr., City of Boulder Colo. 1909–10, County Surveyor, Boulder Co., Colo. 1911, Engr., Water users District #6, Colo. 1912, Engr., Boston Colorado Power Co., Colo. 1913, Engr., Welington Mines, Nevada. 1914, Engr., Nona Mines, Leadville. 1915, Engr., Yellow Pine and Logan Mines, Boulder. 1916, Engr., Chief Engr. for Boulder Tungsten Production Co. Cons. Engr., Tungsten mountain mine, Logan. Yellow Pine and Nona group. Also County Surveyor, Boulder County and U. S. Mineral Surveyor.
Present position: Pres. and Genl. Mgr., The Fred A. Fair Engrg. Assn.

T. H. Powers Farr, Jr., Jackson, Cal.
Proposed by Bradley Stoughton, Benjamin B. Lawrence, Charles F. Rand.
Born 1885, Orange, N. J. 1903–07, Princeton Univ., A. B. 1907–09, Columbia School of Mines. 1909, Aug. and Sept., American Smelters Securities Co., Velardina, Mexico. 1909–10, Engr., American Smelters Securities Co., Matehuala, San Luis Potosi. 1911–14, Examination and development work for private individuals in Ecuador and Southern Colombia, South America.
Present position: Argonaut Mining Co.

John Brainard Fidlar, Tulsa, Okla.
Proposed by H. F. Wright, W. E. Hopper, Irving Perrine.
Born 1892, Davenport, Iowa. 1916, Iowa State College, B. S. Besides the four-year course in mining engineering, I received a year of graduate work in geology but did not write a thesis necessary for a Master's degree. 1915, Iowa Geological Survey.
Present position: Geol., Gypsy Oil Co., working in the Oklahoma fields.

Royce Albert Field, Newcastle, N. S. W., Aust.
Proposed by David Baker, G. D. Delprat, E. J. Horwood.
Born 1889, Lebanon, Pa. Public Schools, Lebanon, Pa. 1905–09, Lebanon High School. 1909–13, Pennsylvania State College, B. S. in Engrg. 1908–12, Summer work, Lebanon Furnaces, Pennsylvania Steel Co. 1913–15, Blast Furnace Dept., Youngstown Sheet & Tube Co.
Present position: 1915 to date; Furnace Foreman, Blast Furnace Dept., Broken Hill Proprietary Co., Ltd.

Martin Fishback, El Paso, Texas.
Proposed by Kirby Thomas, W. G. Swart, D. W. Shanks.
Born 1870, Gudum, Struer, Denmark. 1876–90, Grade and High Schools in Denmark and special instruction in polytechnics. 1893–96, worked in the mines in Colorado while making a special study of geology and mineralogy. 1896–99, Asst. Mgr., Modoc Cons. Gold Min. Co., Boulder Co., Colo. 1899–1900, operated in Crip-

underground experience, Vinegar Hill Zinc Co. 1914 to date, underground and surface workings, Century Zinc Co., Joplin, Mo.
Present position: Asst. Supt., Century Zinc Co.

J. Gary Lindley, Douglas, Ariz.
Proposed by J. Owen Ambler, H. A. Clark, Carl H. Cole.
Born 1888, Moberly, Mo. 1903–06, Moberly High School, Moberly, Mo. 1906–07, Hemet High School, Hemet, Cal. 1908–09, Univ. of Oregon, Eugene, Ore. 1910–13, Univ. of Arizona, Tucson, Ariz., B. S. in Metallurgy. 1910, summer, Chainman, U. S. General Law Office Survey. 1911, summer, Custom Assayer, E. A. Jacobs, Tucson, Ariz. 1912, summer, Mine Assayer with Fred Tatum, Oro Blanco, Ariz. 1913, Conveyor Foreman, C. & A. Smelter, Douglas, Ariz. 1914, Asst. Chemist, C. & A. Smelter, Douglas, Ariz. 1914–16, Engr., National Mines & Smelters Co., Magistral, Dur., Mexico. 1916, Chemist, Douglas Assay Co., Douglas, Ariz.
Present position: Met. Acct., Calumet and Arizona Smelter.

Thayer Lindsley, New York, N. Y.
Proposed by Robert Livermore, Halstead Lindsley, C. Q. Payne.
Born 1882, Yokohama, Japan. 1895–1900, Milton Academy, Milton, Mass. 1901–04, Harvard, A. B. & S. B. 1904–05, Rodman, Rapid Transit Commission, N. Y. 1905–09, Engaged in scouting, and employed in mines, mills, and cyanide plants in Idaho, Colorado, Nevada and Mexico. 1910–12, Exploration and scouting for mines. 1913, Scouting Engr. for the Tonopah Min. Co. 1914–16, Examination and geological work, Goodrich Lockhart Co., New York.
Present position: Min. Geol., Goodrich Lockhart Co.

Robert Davis Longyear, Minneapolis, Minn.
Proposed by E. J. Longyear, John E. Hodge, W. J. Mead.
Born 1892, Petoskey, Mich. 1910–14, Williams College, A. B. 1914–15, Univ. of Wisconsin, M. A. 1915–16, Standford Univ. 1912–14, summer, field geology, E. J. Longyear in Michigan. 1915, Geol., E. J. Longyear Co., Sudbury, Ont.
Present position: E. J. Longyear Co.

Alexander Neil Mackay, Barranquilla, Rep. of Colombia.
Proposed by A. L. Blomfield, T. A. Rickard, Arthur W. Allen.
Born 1877, Aberystwith, England. 1892–95, General education at Eastbourne College, Sussex, England. 1895–99, Technical education at The Royal School of Mines, London, England, graduating as Associate R.S.M. in the division of Mining. 1900, Asst. Assayer, Lake View Cons., Ltd., Kalgoorlie, W. Aust. 1902–04, Head Assayer & Cyanide Mgr., Cosmopolitan Proprietary, Ltd., Kookyine, W. Aust. 1905–07, Asst. Sampler, Asst. Survey Office, Head Sampler, successively, on East Rand Proprietary Mines, Ltd., Transvaal, S. Africa. 1908–11, Asst. Mgr., Cherokee Goldfields, Ltd., San Julian, Chih., Mex. 1912, 1915, Surveyor, Underground Mgr., Genl. Supt., Frontino & Bolivia (S. A.) Gold Min. Co., Ltd., La Salada, Antisquia, Rep. of Colombia, S. Amer.
Present position: Genl. Supt., Frontino & Bolivia (S. A.) Gold Min. Co., Ltd.

Bartlett F. Martin, La Harpe, Kans.
Proposed by E. List, H. A. Buehler, Herbert Taylor.
Born 1881, Shelbyville, Ill. 1898, Common schools. 1899–1903, High School, Shelbyville, Ill. 1904–06, Barnes Med., St. Louis, Mo. 1906–08, University of Missouri. 1908–09, Winona Technical Inst. 1909–11, Chemist, Illinois Steel Co. 1911, Met., Black Hills Smelt. Co. 1911–14, Chief Chem., Hegeler Bros. Zinc Co. 1915, Chemist and Met., U. S. Smelt. Co.
Present position: Chief Chemist, La Harpe, Kans., Zinc Plant, U. S. Smelt. Co.

Sylvester Edward Martin, Wallaroo, So. Aust.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1887, Moonta, So. Aust. 1892–1901, Primary & State School Education. 1901–07, Technical, Mechanical and Electrical Engineering, Surveying, Draftsman's and Chemical Courses, Moonta School of Mines, So. Aust. 1910–14, Technical, Mining Engineering Diploma, International Correspondence Schools. 1901–03, Mechanical Engineering (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt. Co.,

Ltd. 1903–06, Mechanical and Electrical Engineering, Wallaroo & Moonta Min. & Smelt. Co., Ltd. (Wallaroo Smelt. Works). 1906–09, Engineering Draftsman & Surveyor (Wallaroo Smelt. Works), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1909–12, Engineering Draftsman & Surveyor (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1912–14, Engineering Draftsman and Surveyor (Wallaroo Smelt. Works), Wallaroo & Moonta Min. & Smelt. Co., Ltd. 1907–16, Instructor Mechanical and Machine Drawing, Moonta School of Mines, evening classes. 1915 to date: Metallurgical Staff, Wallaroo Smelt. Works.

Present position: Asst. Met., Wallaroo Smelt. Works.

Harry Auld Bell Motherwell, Clifton, Ariz.

Proposed by W. F. Geiger, M. S. Mazany, C. E. Arnold.

Born 1872, Glasgow, Scotland. 1885, Glasgow High School. 1887–89, 1897–98, Glasgow and West of Scotland Technical College, 1890, Mill worker and Chemist, Glengowan Dyeing and Bleaching Works, Scotland. 1899, Chemist, Lake George Copper Mines, N. S. W., Aust. 1901, Chemist, Hercules Gold and Silver Mines, Tasmania. 1902, Chemist, Great Cobar Copper Mines, N. S. W., Aust. 1909, Chemist and Genl. Asst., Scottish-Cardigan Lead Mines, Wales. 1910, Genl. Asst. to Chief Engr., Angola Exploration Syndicate, Angola, S. W., Africa. (Messrs. Farrar Bros., London). 1911, Assayer and Cyanider, Taquah Gold Mine & Exploration Co., Gold Coast, W. Africa. 1912–13, Asst. Engr. and Chemist, Western Orufu Mines, Northern Nigeria, W. Africa. 1914–15, Met. clerk, and mill operator, Inspiration Cons. Copper Co., Ariz. 1915–16, Chemist, International Smelt. Co., Ariz.

Present position: Smelter Chemist, Arizona Copper Co.

Robert Aldridge Nowlin, Elkhorn, W. Va.

Proposed by John J. Lincoln, H. D. Smith, Howard N. Eavenson.

Born 1888, Lynchburg, Va. 1910, Virginia Mil. Inst., B. S. in Civ. Engrg. 1911–12, Student, Civ. Engrg. course, Mass. Inst. of Tech. Several summers prior to 1906, Rodman and Chainman, with Maurice Garland, Lynchburg, Va. 1910, surveying, mapping, building coal washery, reconstructing tipple, etc., responsible for work during absence of Chief Engr., Rodman and Transitman, Freeburn Colliery, Edgarton, W. Va. 1910–11, general mining construction, surveying, office work, plotting surveys, designing small trestles, etc., Transitman and Draftsman, Vest & Cooper, Welch, W. Va. 1911, Draftsman, Pocahontas (Va.) Cons. Collieries Co., principally mine work. 1912, summer, Transitman and Draftsman, C. L. DeMott, Lynchburg, Va. 1913–14, general mining engineering, Glen Alum Coal Co., Glenalum, W. Va.; Transitman for this company and Gilliam, Arlington Shawnee, and Black Wolf Coal & Coke Companies, next to Chief Engr. for these companies; construction work at Glen Alum Coal Co. included opening a new piece of territory, tramway, 830-ft. steel conveyor, steel tipple, railroad tracks, etc., at Gilliam Coal & Coke Co. a picking table was added to old tipple, tracks changed, and walls built.

Present position: 1914 to date; Min. Engr., Crozer-Land Assn.

Harry Fordham Noyes, Newcastle, N. S. W., Aust.

Proposed by H. D. Hibbard, G. D. Delprat, David Baker.

Born 1881, Newton. 1900, Newton High School. 1904, Mass. Inst. of Tech., Dept. of Min. & Met., S. B. 1904–06, Blast Furnace and Bessemer Dept., Maryland Steel Co. 1906–10, Night Supt., New Jersey Zinc Co., Palmerton, Pa. 1910–11, Supt. of Blast Furnace, Princess Furnace Co., Glen Wilton, Va. 1911–14, Genl. Supt., Dayton Coal and Iron Co., Dayton, Tenn.

Present position: Supt. of Blast Furnaces, Broken Hill Prop. Co., Ltd.

Walter S. Palmer, Reno, Nev.

Proposed by George J. Young, J. C. Jones, Bradley Stoughton.

Born 1884, Waterville, Me. 1905, Univ. of Nevada, B. S. in Min and. Met. 1907, Columbia School of Mines, E. M. 1905, Greene Cons. Copper Co., Cananea, Mex. 1906, Mining and Metallurgical practice in the field, summer months, Virginia City, Nev., and Wilkes-Barre, Pa. 1907–08, Mine Surveyor, U. S. Smelt., Ref. and Min. Co., Mammoth, Cal. 1908–09, Asst. Chemist, Balaklala Cons. Copper Co., Coram, Cal. 1909–10, Min. Engr., U. S. General Land Office. 1910–13, Instructor in Mining and Metallurgy, Univ. of Nev. 1910, Chemist and Assayer, Nevada State Mining Laboratory. 1913, Asst. Prof. of Mining and Metallurgy, Univ. of Nev.

Present position: Chemist and Assayer, Nevada State Mining Laboratory and Asst. Prof., Mining and Metallurgy, Mackay School of Mines.

George S. Patterson, Edwards, N. Y.
Proposed by Homer L. Carr, William Campbell, E. J. Hall.
Born 1892, Denver, Colo. 1914, Columbia School of Mines, E. M. 1914–16, underground mining operations and surveying, New Jersey Zinc Co.
Present position: Northern Ore Co.

John Lathern Pearson, Wallaroo Mines, So. Aust.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1871, Alston, Cumberland, England. 1877–84, Attended Church of England Day School and Public Schools, England. 1885–87, Studied Mathematics at a Night School, Moonta Mines, S. Amer. 1891–94, 1898–1901, Attended classes at Moonta School of Mines in Mine Surveying, Mineralogy, Metallurgy, Assaying, Mechanical Drawing, Steam and Practical Chemistry. 1885–89, Employed in Moonta Mining Company's Workshops at Moonta Mines, S. A. 1889–93, underground work at Moonta Mines. 1893–95, Asst. in Survey Dept., Wallaroo & Moonta Min. & Smelt. Co., Wallaroo and Moonta Mines. 1895–98, In charge of prospecting operations at Kalgoorlie, Kanowna and Menzies, Western Aust. 1898–1902, Underground survey work and sectional supervision of general mining work for Wallaroo & Moonta Min. & Smelt. Co. 1902–03, Asst. Mine Mgr., Nth. Lyell Mine, Tasmania. 1903–09, Mine Mgr., Queensland Copper Co., Mt. Perry, Queensland. 1910, Six months as Mine Mgr. at Mt. Jasper Mine, Heazlewood, Tasmania. 1910–15, Asst. underground officer at Wallaroo Mines.
Present position: 1915 to date; Chief underground officer, Wallaroo Mines.

Fred Hough Perkins, Glendale, Ariz.
Proposed by Thomas Armstrong, W. E. Defty, F. W. Libbey.
Born 1876, Kansas City, Mo. 1886–92, Ward School, Kansas City, Mo. 1893–96, High School, Kansas City, Mo. 1897–99, Missouri School of Mines, B. S. in General Science. 1900, Missouri School of Mines, B. S. in Min. Engrg. 1898, summer, L. X. Smith Assay Office, Cripple Creek, Colo., also Rio Grande Sampler of that place. 1899, summer, Perkins & Perkins, Assayers, also Supt., Kopy Kat Lead & Zinc Min. Co., Joplin, Mo. 1900, Asst. Min. Engr., Diamondville Coal and Coke Co., Diamondville, Wyo. 1901, Asst. Engr. and Assayer, Utah Cons., Bingham Canyon, Utah. 1902–03, Cons. Engr., Red Wing and Utah Apex Min. Cos., Bingham Canyon, Utah. 1904, General Min. Engrg. specializing in mine litigation. 1905, Min. Engr., Blue Ledge Min. Co., Jacksonville, Ore. 1907–08, County Engr., Jackson Co., Ore. 1909–15, General practice of Mine Engrg. in Arizona.
Present position: 1915 to date; Genl. Mgr. Shamrock Cons. Min. Co., Castle Hot Springs, Ariz.

Ocha Potter, Houghton, Mich.
Proposed by A. H. Wohlrab, F. W. Paine, Reginald E. Hore.
Born 1878, Sherwood, Wis. 1911, Michigan College of Mines, B. S. 1905–09, Mgr., Houghton-Alaska Exploration Co.
Present position: 1909 to date; Supt., Superior Copper Co. and Chief Efficiency Engr., Calumet & Hecla and associated mines.

Walter C. Rea, Ruth, Nev.
Proposed by H. L. Handley, George R. Hilby, W. S. Larsh.
Born 1893, Gallatin, Mo. 1913, Preparatory work, Wentworth. 1914, Preparatory work, Military Academy, Lexington, Mo. 1914, Mackay School of Mines, Univ. of Nevada, Reno, Nev. 1914–15, Surveyor, Buckhorn Mines Co., Buckhorn, Nev. 1915, Met. Experiments, Golden Cycle Mill, Colorado Springs, Colo. 1915 to present, General Engineering and Chemist, Nevada Cons. Copper Co., Ruth, Nev.
Present position: Engr., Ruth Mine.

Donald Duryea Reed, Kadina, South Australia.
Proposed by H. Lipson Hancock, William E. Slee, Albert L. Brown.
Born 1890, Wallaroo Mines, South Australia. 1895–1904, Public School, Wallaroo Mines. 1904–06, District High School, Moonta. 1906–10, Technical—School of Mines, Moonta, Depts. of Drawing, Chemistry, Assaying & Mineralogy. 1906–11, Met. Assay Dept. (Wallaroo Smelt. Works) & Chemical Laboratory (Wallaroo Mines), Wallaroo & Moonta Min. & Smelt Co., Ltd. 1911–16, Assisting in Supervising at Converter & Electrolytic Departments. 1915 to date: Mathematical Instructor, Moonta School of Mines, evening classes.
Present position: Asst. Met., Wallaroo Smelt. Works.

Charles Ramsay Rinehart, Allentown, Pa.
Proposed by F. L. Antisell, S. Skowronski, Edward Heller.
Born 1875, Uniontown, N. J. 1899, Grad., Lafayette College, Civ. Engr. Tippett & Wood, Phillipsburg, N. J. Great Northern Portland Cement Co., Marlboro, Mich. Hudson Portland Cement Co., Hudson, N. Y. Seaboard Portland Cement Co., Alsen, N. Y. Lehigh Car, Wheel and Axle Works, Catasauqua, Pa. Fuller Engineering Co., Allentown, Pa. Lehigh Foundry and Lehigh Stoke Co., Catasauqua, Pa.
Present position: New York Representative, Lehigh Car, Wheel and Axle Works, Catasauqua, Pa.; Fuller Engineering Co., Allentown, Pa.; Lehigh Foundry and Lehigh Stoke Co., Catasauqua, Pa.

Bernard T. Rocca, Mayer, Ariz.
Proposed by J. L. White, J. N. D. Gray, A. F. Bassett.
Born 1892, Great Western Q. S. Mine, Lake Co., Cal. 1915, Univ. of Cal., B. S., College of Mining. 1914, General work, surface and underground, Helen Quicksilver Mine, Lake Co., Cal. 1915 to date, surveying, assaying, draughting, Cons. Arizona Smelt. Co., Humboldt, Ariz.
Present position: Engr., Bluebell Mine, Cons. Ariz. Smelt. Co.

C. Floyd Sapper, Herculaneum, Mo.
Proposed by William Allen Smith, Franz Cazin, S. Paul Lindau.
Born 1886, St. Louis, Mo. Grammar School, St. Louis. 1908–10, Univ. of Missouri, Mech. Engrg. 1910–15, Univ. of Missouri, specialized in chemistry and allied subjects. Substituted credit from Univ. of Missouri for high school credit. 1904–06, Mill Machine and Tool Co., St. Louis. 1906–08, Engrg. Dept., Century Electric Co., St. Louis. 1911, summer, Mech. Dept., Curtis & Co. Mfg. Co., St. Louis. 1912–13, summers, Drafting, Herculaneum plant, Engrg. Dept., St. Joseph Lead Co.
Present position: 1914 to date, Engrg. Staff, St. Joseph Lead Co.

Matthew Johnston Scammell, Sparrows Point, Md.
Proposed by Quincy Bent, Charles F. Rand, Edgar C. Felton.
Born 1883, Trenton, N. J. 1900, High School, Trenton, N. J. 1905, Lafayette College, Easton, Pa., B. Sc. Roebling Sons' Co., Trenton, N. J. Republic Iron & Steel Co., Youngstown, O. Carnegie Steel Co., Youngstown, O. Maryland Steel Co., Sparrows Point, Md.
Present position: Genl. Supt., Maryland Steel Co.

Anton Schneider, Brewster, Polk Co., Fla.
Proposed by David Taylor, Benjamin L. Miller, George W. Engel.
Born 1871, Summit Hill, Pa. 1892, Lehigh Univ., C. E. 1892–98, Union Pacific R. R. 1898–99, U. S. Engineers at Honolulu. 1899–1902, Manhattan Ry. Co., New York. 1902–04, Cerro de Pasco Co., Peru, So. Amer. 1904–05, Alphons Curtis Chimney Const. Co. 1905–08, Div. Engr., East River Tunnels, Rapid Transit Subway Const. Co., New York. 1908–11, Supt.,Pierce Phosphate Co., Fla.
Present position: 1911 to date; Supt., Amalgamated Phosphate Co.

Arthur S. Shoffstall, New Brighton, Staten Island, N. Y.
Proposed by Robert S. Stanley, Walter O. Snelling, E. T. McCleary.
Born 1876, Jefferson Co., Pa. 1895, Public Schools, Brookville, Pa. 1895–96, Preparatory Dept., The Pennsylvania State College. 1896–1900, The Pennsylvania State College, School of Natural Science, B. S. 1905, M. S. 1900–01, Asst. in the Chemical Laboratories, The Pennsylvania State College. 1903–05, Instructor, Dept. of Chemistry, The Pennsylvania State College. 1905–07, Supt., Sulphuric Acid Plants, E. I. duPont Powder Co., Gibbstown, N. J. 1907–08, Supt., Acid Plants, Naval Proving Ground, Indian Head, Md. 1908–10, Technical Staff, Orford Copper Co., Bayonne, N. J. 1910–13, Supt., Sheet Mill Dept., The International Nickel Co., Bayonne, N. J.
Present position: Research Engr., The International Nickel Co., Bayonne, N. J.

Ellsworth H. Shriver, Gillham, Sevier Co., Ark.
Proposed by George A. Morrison, C. W. Whitley, J. H. Frantz.
Born 1890, Washington, Pa. 1909, Martins Ferry High School. 1914, Engr. of

Mines, The Ohio State Univ. 1911, Genl. Engrg. with W. C. Fawcett, C. E., Martins Ferry, O. 1911–12, Asst. County Engr., Belmont Co., O. 1913, Genl. Min. Engrg., Bellaire, O. 1914, Investigation Coal Lands, East Ky., for Major J. A. Kirkpatrick, Springfield, O. 1914–15, Asst. Dept. Mine Eng., The Ohio State Univ. 1915–16, Investigation Mineral Lands and Water Power Sites, The Ohio State Univ.

Present position: 1916 to date; Met., The American Star Antimony Co.

Daniel Sillars, Yorkshire, England.

Proposed by W. R. Walker, J. E. Stead, William Thomlinson.

Born 1879, Glasgow, Scotland. Three years apprenticeship with Messrs. R. R. Gatlock and Thomson, City Analysts of Glasgow. Two years Metallurgy, Glasgow & West of Scotland Technical College, Senior Prizeman. Eight years Metallurgical Chemist, William Beardmore & Co., Ltd., Parkhead Forge, Glasgow. Five years, Chief Chemist, Messrs. Seaton Carew Iron Co., Ltd., West Hartlepool.

Present position: Chief Chemist, Messrs. Bolckow Vaughan & Co., Ltd., Cleveland Iron & Steel Works, South Bank S. O., Yorkshire.

Laurence Heck Stone, Easton, Pa.

Proposed by Ralph H. Sweetser, Harry P. Sweeny, W. Spencer Hutchinson.

Born 1889, Easton, Pa. 1909, Grad., Easton High School. 1913, Grad., Lafayette College, E. M. 1911, summer, Machinist Helper, Erection shop, Ingersoll-Rand Co., Philipsburg, N. J. 1912, summer, Rod and Chain, Genl. Mgrs. Corps, Lehigh Valley R. R. 1913, Transitman and Surveyor, Wilkes-Barre Anthracite Coal Co., Wilkes-Barre, Pa. 1913–14, Asst. Mine Surveyor, Thomas Iron Co., Richard Mine, Wharton, N. J. 1914–15, Mine Surveyor and Engineer, Thomas Iron Co., Richard Mine, Wharton, N. J.

Present position: 1915 to date; Asst. Supt., and Engr., Thomas Iron Co., Richard Mine.

Charles H. Thole, Rye Valley, Ore.

Proposed by L. A. Walker, Frank W. Parker, A. W. MacNichol.

Born 1878, Dwight, Ill. 1896, High School. 1896–98, So. Dakota School of Mines. 1898–1902, underground, Homestake Min. Co., Lead, S. D. 1905–06, Mine Foreman, Desert King Min. Co., Ballarat, Cal. 1907, underground, City of Los Angeles, Engrg. Dept. 1908, Leasing, Aurora, Nev. 1909–10, Sampler, Shift boss, Assayer, Ruby Gulch Min. Co., Zortman, Mont.

Present position: 1912 to date; Mill Foreman, Rainbow Mine.

James Ezra Thoms, Depue, Ill.

Proposed by Robert H. Richards, W. M. Kelsey, D. C. Wray.

Born 1883, Three Rivers, Mich. Public Schools. 1905–06, Michigan College of Mines. 1908–10, Engr., Live Oak Dev. Co., Globe, Ariz. 1910–13, Asst. Supt., U. S. Gypsum Co., Gypsum, O.

Present position: 1913 to date; Asst. Chief, Milling and Roasting Dept., Mineral Point Zinc Co.

Adolph F. Zang, Denver, Colo.

Proposed by Irving T. Snyder, D. W. Brunton, A. E. Carlton.

Born 1890, Denver, Colo. 1913, Cornell Univ., A. B.

Present position: Director and Asst. Treas., The Vindicator Cons. Gold Mining Co.; Secy., The Cresson Cons. Gold Min. & Mill. Co., both of Cripple Creek, Colo. In Bond Dept., The German American Trust Co., Denver.

Associate Members

Walter S. McKee, Chicago, Ill.

Proposed by A. C. Ludlum, Newton Cleaveland, Charles Janin.

Born 1876, Watervleit, Mich. 1883–87, Grammar School, St. Charles, Ill. 1887–96, High School, Browning, Mo. 1900, International Correspondence School, Course in drafting. 1897–99, Asst. Cashier, Citizens Bank, Aledo, Ill. 1899–1900, Asst. credit man, A. G. Spalding & Co., Chicago. 1900–01, In the accounting department, The Sargent Co., Chicago. 1901–02, Salesman, The Sargent Co., Chicago. 1902–10, Sales Mgr., Steel Foundry Dept., American Brake Shoe and Foundry Co., Chicago. 1910–11, Genl. Sales Mgr., American Manganese Steel Co., Chicago.

Present position: 1911 to date; Vice-Pres., American Manganese Steel Co.

Carlos Enrique Michels, Huanuni, Bolivia, So. Amer.
Proposed by William Braden, Henry G. Ferguson, Ira B. Joralemon.
Born 1889, Lima, Peru. 1904, Institute de Humanidades, Santiago, Chile. 1906–07, Draughtsman, Braden Copper Co., Chile. 1907–09, Asst. to Chief Engr., Braden Copper Co., Chile. 1910–12, Met. Dept., Braden Copper Co., Chile. 1913–14, Asst. Supt., Llallagua Tin Mines, Bolivia.
Present position: 1915 to date; Supt., Met. Establishment and Mill, Simon Patino Tin Mines.

T. E. Snyder, Hazelton, Pa.
Proposed by A. B. Jessup, Charles Enzian, J. M. Humphrey.
Born 1857, St. Johns, Pa. Midvalley Coal Co., Wilburton, Pa. Maryd Coal Co., Maryd, Pa. J. S. Wentz & Co., Hazle Brook, Pa. Upper Lehigh Coal Co., Upper Lehigh, Pa.
Present position: General Manager.

Roger W. Straus, New York, N. Y.
Proposed by Judd Stewart, P. A. Mosman, Willard S. Morse.
Born 1891, New York. 1913, Princeton, Litt.
Present position: Asst. to Chairman, Board of Directors, American Smelt. & Ref. Co.

Gustave C. Taussig, St. Louis, Mo.
Proposed by George O. Carpenter, Herman Garlichs, Arthur P. Watt.
Born 1884, St. Louis, Mo. 1902, Grad., St. Louis High School.
Present position: Asst. Secy., St. Louis Smelt. & Ref. Co.

Frank J. Gustin, Salt Lake City, Utah.
Proposed by Ernest Gayford, C. W. Whitley, C. W. Stimpson.
Born 1877, Princeton, Ill. 1897, L. L. B., University of Nebraska. Member of American Bar Association; Utah State Bar Association; American Institute of Criminal Law. Interested directly and indirectly with a number of mining companies. Member of the Bar of the States of Nebraska, Utah, Nevada, Idaho and United States Supreme Court.
Present position: Attorney at Law. Member of the firm of Gustin, Gillette & Broyton.

Junior Members

Harlowe Hardinge, Ithaca, N. Y.
Proposed by H. W. Hardinge, R. B. T. Kiliani, Ronald Clark.
Born 1894, Denver, Colo. 1910–12, Attended Tome School, Port Deposit, Md. Have been on a considerable number of extensive trips and have assisted in the operation of Hardinge conical mills at a number of plants.
Present position: 1912 to date; Sibley College, Cornell Univ.

Richard Paul Ernst Hermsdorf, New York, N. Y.
Proposed by William Campbell, E. J. Hall, Charles P. Berkey.
Born 1895, New York, N. Y.
Present position: Student, Met. Engrg., Columbia Univ.

Charles Francis Williams, Chivatera, Son., Mex.
Proposed by John A. McDonald, A. S. Konselman, W. A. Richelsen.
Born 1894, Mansfield, O. 1914, Grad., St. John's Military Academy, Delafield, Wis. 1914 and 1915, New Mexico State School of Mines. 1915, Colorado School of Mines. 1916, Asst. to Met. Engr., Chino Copper Co., Hurley, New Mexico.
Present position: Mine Engr., Cananea Cons. Copper Co.

Change of Status—From Associate Member to Member

Edmund Mozier King, Leura, Launceston, Tasmania.
Born 1848, Launceston, Tasmania. 1889–1902, College Saint Croix, Aux. terms, Paris. 1902–06, Church of England Grammar School, Launceston, Tasmania.

Studied Chemistry under R. N. Hobart. 1887 to date: Director, Mt. Bischoff Tin Min. Co. 1886–1915, Director, Tasmania Gold Min. and its reconstruction until liquidated. 1897 to date: Director, Tasmania Copper Co. (at present Chairman) situated at Rosebery, West Coast of Tasmania. Foundation member of the Australian Institute of Min. Engrs.

Present position: Director of various mining companies.

From Junior Member to Member

William Cristy Pryer, Anaconda, Mont.

Proposed by C. R. Wraith, Enoch A. Barnard, Albert E. Wiggin.

Born 1890, Falmouth, Mass. 1914, So. Dak. School of Mines, B. S. 1914, Assayer, Titanic Min. Co., Deadwood, S. Dak. 1915, Prospect examination with S. R. Thompson, Deadwood, S. Dak. 1916, Testing Dept. and Chemist, Anaconda Copper Min. Co.

Present position: Chemist, Anaconda Copper Min. Co.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Aug. 1, 1916 to Sept. 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 116, February to September, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Sept. 10, 1916.

Ahlers, R. O., 1–4 Norfolk House, Laurence Pountney Hill, London, E. C., England.
Allen, Arthur Watts........................Instructed to hold everything.
Ambler, Harry A........................Box 540, Rolla, Mo.
Anderson Robert John, Asst. to Prof. Henry M. Howe, Broad Brook Road, Bedford Hills, N. Y.
Applegate, J. S...........Material Engr., Wagner Electric Mfg. Co., St. Louis, Mo.
Atherton, T. W. T........................Kagesman, Kars., Obi, Russia.
Bassett, Thomas E........................236 Park St., Pontiac, Mich.
Belden, S. B., Vice-President, The Jeffrey Manufacturing Co., Columbus, Ohio.
Benham, W. M...............Co. H, 1st Ariz. Infantry, Fort Huachuca, Ariz.
Bicknell, Harold Lewis........................Hennessy Annex, Butte, Mont.
Billick, Don C........................Box 99, Alturas, Cal.
Bonsib, R. S...................U. S. Metals Refining Co., East Chicago, Ind.
Boydell, H. C., Prestea Block A., via Seccondee, Gold Coast Colony, W. Africa.
Bradley, Henry F...............11 Bransford Apartments, Salt Lake City, Utah
Bridge, Samuel D., Care Patricio Milmo e Hijos, Sucs., Banqueros, Monterrey, Mex.
Buell, L. T..........Ore Trading Co., Ltd., Casilla 851 Antofogasta, Chile.
Callen, A. S........................Hold publications
Careaga, C. R........................Calle de la Florida 5, Seville, Spain.
Carpenter, Edwin Leon........................Box 397, Salisbury, Md.
Carstens, Carl Eberhard........................400 W. 4th St., Anaconda, Mont.
Chatin, August H........................Box 424, Walsenburg, Colo.
Claghorn, James L........................Forks of Salmon, Siskiyou Co., Cal.
Clapp, C. H., Prof. of Geol.......Montana State School of Mines, Butte, Mont.
Clarke, Edgar W........................622 Second Ave., Johnstown, Pa.
Coe, Ira J........................362 Euclid Ave., Oakland, Cal.
Cole, A. Newton.........H. Koppers Co., Care United Furnace Co., Canton, O.
Cunningham, Noel........................200 5th Ave., New York, N. Y.
Cushing Daniel........................17 Fairview St., Waterbury, Conn.
Dammann, A. C........................5410 Wayne Ave., Chicago, Ill.
Davis, John A........................U. S. Bureau of Mines, Washington, D. C.
Dickman, Robert N........................170 Bay St., St. Augustine, Fla.
Doyle, John J..................Cosden Oil & Gas Co., Drawer P, Tulsa, Okla.
Du Moulin, Walter Louis, Mech. Engr., Andes Copper Mining Co., Chanaral, Chile, South America.
Dwyer, C. Eustace...............Federal Min. & Smelt. Co., Wallace, Idaho.

Dye, A. V., Asst. Genl. Mgr.................Phelps, Dodge & Co., Douglas, Ariz.
Eames, Luther B.........Hollinger Gold Mines, Ltd., Timmins, Ont., Canada.
Eldredge, Robert B.......................Ray Cons. Copper Co., Ray, Ariz.
Emery, William, Jr...Natalie, Pa.
Emmons, N. H., Mgr..............Tennessee Copper Co., Copperhill, Tenn.
Filteau, Charles A............St. Lawrence Talc Co., Natural Bridge, N. Y.
Fisher, Howell T.........................6 Spring Lane, Englewood, N. J.
Foerster, Hallard W., Care Tigre Mining Co., Esqueda, Son.,
Mexico, via Douglas, Ariz.
Fox, Walter V............................Coralbut Mining Co., Joplin, Mo.
Fuketa, F.,.........Mitsubishi Kaisha, Mining Dept., Marunouchi, Tokyo, Japan.
Ganthier, C. B................................P. O. Box 252, Tucson, Ariz.
Gee, J. Emerson...................304 H. W. Hellman Bldg., Los Angeles, Cal.
Grant, Bertram........................615 Market St., Wilmington, Del.
Greit, Adolph Julius....................Calle 13 Y 6, Vedado, Habana, Cuba.
Gross, John, Mining and Metallurgical Engr., 423 McPhee Bldg., Denver, Colo.
Grubnau, V. Carl.........................114 Arch St., Philadelphia, Pa.
Guernsey, John B............................132 W. 4th St., New York, N. Y.
Guiterman, Kenneth S., Selby Smelting & Lead Co.,
804 Merchants Exchange Bldg., San Francisco, Cal.
Haldeman, G. T..Hold publications.
Hamilton, C. W., Mexican Gulf Oil Co., Apartado 106, Tampico, Tamps., Mex.
Hamilton, W. R., Petroleum & Min. Engr., Hobart Bldg., San Francisco, Cal.
Hance, James H.....................815 North Johnson St., Iowa City, Iowa.
Harrington, Daniel, U. S. Bureau of Mines, 334 Post Office Bldg., Butte, Mont.
Hastings, John B...............................P. O. Box 954, McGill, Nev.
Head, James L.............Care Calumet & Arizona Min. Co., Warren, Ariz.
Hill, Walter Hovey............................917 Franklin St., Boise, Idaho.
Hodgson, Joseph P., Cons. Engr., Mining Dept., Phelps, Dodge & Co., Bisbee, Ariz.
Hoffman, Lloyd...Pottersville, N. J.
Holderer, George B.........General Chemical Co., 25 Broad St., New York, N. Y.
Holland, L. F. S................Care Cons. Arizona Smelt. Co., Humboldt, Ariz.
Hollister, Scovill E............Care Boston-Corbin Copper Co., Corbin, Mont.
Holmes, George F............................140 Ogden Ave., Menominee, Mich.
Honnold, W. L., Care The Commission for Relief in Belgium,
120 Broadway, New York, N. Y.
Hoover, Theodore J..Palo Alto, Cal.
Hotta, M...................136 Sumiyoshi, Mukogun, Hyogo Pref., Japan.
Hu, S. H.....................Hartley Hall, Columbia Univ., New York, N. Y.
Hutchins, J. P.............Morskaya 21, Apartment 24, Petrograd, Russia.
Hyde, James M..........................635 Mills Bldg., San Francisco, Cal.
Jackson, G. R., Supt. Negaunee Dist., Cleveland-Cliffs Iron Co., Negaunee, Mich.
Jahn, William F., Mill Supt., Tough-Oakes Gold Mines, Ltd.,
Kirkland Lake, Ont., Canada.
Jenks, T. H., Genl. Mgr. Gold Bar Mines Co.................Wickenburg, Ariz.
Jenney, Warren, Comptroller, Andes Copper Mining Co., Chanaral,
Chile, South America.
Jensen, Joseph, Dept. of Metallurgy and Mining,
Carnegie Institute of Technology, Pittsburgh, Pa.
Johnson, J. Harlan, Care Sanitary Troop, 4th So. Dak. Infantry, San Benito, Tex.
Jones, Thomas J.......Care Kyshtim Co., Nevsky Prospect 1, Petrograd, Russia.
Jordan, R. D.................................604 C. Street, Sparrows Point, Md.
Kamiyama, Tatsuzo, 444, Nakamaru, Kami osaki, Yebaragun, Tokyfu, Japan.
Keeney, Robert M..............The Ayres Hotel, 1441 Logan St., Denver, Colo.
Kelley, Arthur, L., Supt............American Tungsten Co., Tucson, Ariz.
Kepner, R. B...Humboldt, Ariz.
Koch, Arthur W.................Granby Mining & Smelting Co., Granby, Mo.
Krebs, Joseph J...Bisbee, Ariz.
LaCroix, Morris F...........................243 Ocean St., Lynn, Mass.
Lavery, V. M...Hold publications.
Lee, Montrose L.......Care Clark, Dodge & Co., 51 Wall St., New York, N. Y.
Lewis, Robert S...................University of Utah, Salt Lake City, Utah.
Lindemuth, L. B.....................Pennsylvania Steel Co., Bethlehem, Pa.
List, Elmer..........................Care U. S. Smelt. Co., La Harpe, Kans.
Loo, P. C...................................1075 Boylston St., Boston, Mass.
Lovejoy, John M.............Geol. Staff, Producers Oil Co., Billings, Mont.

Low, V. F. Stanley, Care National Bank of Australasia, 5 Bishopsgate,
London, E. C., England.
Lunn, Robert, Jr. .... Instructed to hold all mail.
McBride, Wilbert George .... P. O. Box 816, Morenci, Ariz.
McDowell, J. Spotts, Harbison-Walker Refractories Co.,
1705 Farmers Bank Bldg., Pittsburgh, Pa.
McHugh, P. M., Mgr., The Dorr Machine Co., 812 Cooper Bldg., Denver, Colo.
Macfarlane, Rienzi W. .... Morenci Hotel, Morenci, Ariz.
MacGowan, John K .... 190 Riverside Drive, New York, N. Y.
Madson, Frank H .... Box 708, Bessemer, Mich.
Maller, John W .... 303 E. 161st St., New York, N. Y
Mansfield, M .... 860 East 17th So. St., Salt Lake City, Utah.
Marshall, Holman T. .... Care Dutchman Lease, American Fork, Utah.
Marshall, Stewart McC., Chief Engr., Southwark Foundry & Machine Co.,
Philadelphia, Pa.
Maxon, Walter Landon .... Winthrop, Shasta Co., Cal.
Merryweather, Hubert, Bethlehem Chile Iron Mines Co., La Higuera,
Coquimbo, Chile, South America.
Miller, Harry H .... 803 Dos Robles Drive, Alhambra, Cal.
Millard, William J .... 818 House Bldg., Pittsburgh, Pa.
Mitchell, George .... 1367 S. Figueroa St., Los Angeles, Cal.
Montoulieu, Eduardo I .... Jovellar 27 (altos) Havana, Cuba.
Moore, Redick R .... Instructed to hold everything.
Muir, Downie D., Jr., Intermountain Mgr., United States Smelt., Ref. & Min.
Exploration Co., P. O. Box 1785, Salt Lake City, Utah.
Najarian, H. K .... Missouri Cobalt Co , Fredericktown, Mo.
Neeld, Harper C .... Weedon Mining Co., Weedon, Quebec, Canada.
Newell, George S .... 742 E. Pennsylvania Ave., San Antonio, Tex.
Nye, Robert .... Nevada City, Cal.
O'Harra, C. C., Pres. and Prof. of Geol., South Dakota State School of Mines,
Rapid City, S. D.
Ord, James .... Levack, Ont., Canada.
Pallansch, Rollin A .... Hold publications.
Parmelee, H. C .... Colorado School of Mines, Golden, Colo.
Pearl, Holman I .... Rice Lake, Wis.
Peterson, Clarence J .... B. P. O. Elks Club, San Francisco, Cal.
Pfouts, Elmer .... Kelly, New Mexico.
Phalen, William C., Mineral Technologist, U. S. Bureau of Mines, Washington, D. C.
Place, Richard G .... Colonial Hotel, Reno, Nev.
Ravicz, Louis .... 854 McKnight Bldg., Minneapolis, Minn.
Reifsneider, Le Baron B .... 32 So. 34th St., Camden, N. J.
Rhodes, C. E .... R. D. 1, Box 363, Pasadena, Cal.
Riddell, Guy C., Cons. Met., Broken Hill Associated Smelters Proprietary, Ltd.,
Port Pirie, So. Aust.
Robbins, Hallet R.. .... 814 Birks Bldg., Vancouver, B. C., Canada.
Rodgers, Joseph H .... 1267 Laurel Ave., Los Angeles, Cal.
Ryder, Thomas J .... Apartado 1203, Mexico City, Mexico.
Schroter, George A ...... W. C. Laughlin Co., 115 Broadway, New York, N. Y.
Sanders, Bernard H .... 1 Penventon Terrace, Redruth, Cornwall, England.
Saxman, R. M .... Villa Nova, Pa.
Scheble, M. C .... Min. Engr., P. O. Box 867, San Antonio, Tex.
Scott, J. Ralph .... Feldspars, Ltd., Hartington, R. R. No. 1, Ontario, Canada.
Sherman, Gerald F. G., Civ. and Min. Engr., Supt. Mine Dept., Copper
Queen Cons. Mining Co., Bisbee, Ariz.
Sherry, H. K .... Platteville, Wis.
Smith, Frank A .... 279 10th St., San Pedro, Cal.
Smith, Lyon .... E. R. Smith & Co., 77 Bedford St., Boston, Mass.
Smither, Thomas McC .... 429 Walnut St., Reno, Nev.
Soper, R. H .... 308 North Topka Ave., Wichita, Kans.
Spearman, Charles, The Renfrew Molybdenum Mines, Ltd., Mt. St. Patrick,
Ont., Canada.
Sproat, A. D .... P. O. Box 388, Bound Brook, N. J.
Stack, James R .... American Smelt. & Ref. Co., Maurer, N. J.
Stephenson, Francis Lewis .... 1117 Columbia Ave., Scranton, Pa.
Stewart, M. B .... Metropolis, Ill.
Tillman, Charles A .... 320 McCormick Bldg., Chicago, Ill.

Stroup, Thomas A............No. 3 Louise Apartments, Salt Lake City, Utah.
Suverkrop, Edward A..................................Box 724, Dayton, O.
Suydam, A. G..........................1325 Second St., Santa Monica, Cal.
Tarantous, Richard, Care C. S. Herzig, Cons. Engr.,
27 William St., New York, N. Y.
Thompson, Warren D......Min. Engr., Nevada Cons. Copper Co., Ruth, Nev.
Tibby, Benjamin F.........................178 E. St., Salt Lake City, Utah.
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1891	*Towne, Robert S.	Aug. 3, 1916.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Manganese Ores of the Lafayette District, Minas Geraes, Brazil

BY JOSEPH T. SINGEWALD, JR.,* PH. D., BALTIMORE, MD., AND BENJAMIN LEROY MILLER,† PH. D., SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1917)

INTRODUCTION

FOR a number of years Russia, India and Brazil have outranked all other countries as producers of manganese ores. During the 5 years immediately preceding the European war, the average annual production of Russia was 844,000, of India 694,000 and of Brazil 200,000 long tons. Since the outbreak of the war there has been a considerable falling off in the Russian and Indian production, particularly in the former, whereas the production of Brazilian ores has been greatly increased, amounting in 1914 to over 250,000 tons and in 1915 to nearly 350,000, with conditions favorable for a still larger production in 1916. Manganese is one of the few industrially important metals that are not produced in the United States in quantity commensurate with our needs, so that we have been compelled to import annually about 300,000 tons, having a value of over $2,000,000, and these ores have been obtained from the three countries mentioned above. Their rank as contributors to our imports of manganese ores was India, Russia, Brazil; and in 1913, Brazil contributed only one-fifth. During the past 2 years Brazil has furnished rapidly increasing quantities to this country, and, with the falling off of imports from India and Russia, has become our principal foreign source. In 1914, Brazil furnished two-fifths of the imports and more than India or Russia, and in 1915 over nine-tenths. The manganese ores of Brazil are consequently of more than usual interest to us at this time.

THE MANGANESE-ORE DISTRICTS OF BRAZIL

The manganese mining industry of Brazil dates from the year 1894, and since that year the total production has been over 3,000,000 tons. With the exception of a small quantity produced in the State of Bahia from deposits west of the City of Bahia, this output has come from the

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State of Minas Geraes and has been exported through the port of Rio de Janeiro. In Minas Geraes there are two distinct, though not widely separated, manganese districts, known generally as the Miguel-Burnier and the Queluz or Lafayette districts. The Miguel-Burnier district was the first to be developed and for a number of years was the more important producer; but, since the phenomenal development of the Morro da Mina mine, the Lafayette district has assumed far greater importance.

In order to contrast the modes of occurrence of the deposits in the two districts, a brief description of the Miguel-Burnier district will be given, in the nature of a summary of a previously published fuller account.[1] The deposits of this district extend as a narrow belt about 10 miles long, parallel to the Ouro Preto branch of the Estrada da Ferra Central do Bresil, and lie at the southern edge of the great iron-ore region of central Minas Geraes. The orebodies occur as lenses or beds intercalated in a series of sedimentary strata showing a rapid succession of itabirite, schist, calcareous schist, and limestone. Stratigraphically they are found in the upper part of the Itabira iron-formation and in the lower part of the overlying Piraçicaba schist,[2] formations of probable Algonkian age. The ores are very high-grade manganese oxides, chiefly a mixture of psilomelane and pyrolusite, averaging 50 per cent. manganese, 1 per cent. silica, and 0.03 to 0.05 per cent. phosphorus. The great drawback of the district is the fact that the beds are steeply dipping and narrow, rarely over 6 ft. in width, so that expensive underground mining must be resorted to, and the individual deposits are relatively small. Two views have been advanced to explain the origin of the ores. H. K. Scott, who has written the most complete account of them, says:[3] "Whatever may have been the original state of the manganese ore bed, there can be no doubt that in its present condition, and down to the level to which it has been worked, it is a residual deposit from which the other elements have been leached out." O. A. Derby, in a discussion of Scott's paper, endorsed this view and referred to the original state of the manganese ore beds as limestone with varying proportions of metallic carbonates and siliceous impurities.[4] On the other hand, Harder and Chamberlin say:[5] "From their occurrence it must be assumed that they are similar in origin to the associated rocks, that is, that they are original sedimentary deposits of manganese oxide." The same statement is made by Harder alone.[6] Our own feeling is that while one cannot positively state that the ores were not laid down in the form of manganese oxides as integral parts of a sedimentary series, their relations to the associated limestones are such as to make their interpretation as residual products of decomposition and replacement of manganiferous limestone the more probable explanation of their origin.

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[1] Joseph T. Singewald, Jr., and Benjamin LeRoy Miller: High-Grade Manganese Ores of Brazil. *The Iron Age*, vol. 97, pp. 417–420 (1916).

[2] E. C. Harder and R. T. Chamberlin: Geology of Central Minas Geraes, Brazil. *Journal of Geology*, vol. 23, pp. 358–363 (1915).

E. C. Harder: Manganese Ores of Russia, India, Brazil and Chile. *Bulletin* No. 113, p. 788 (May, 1916).

[3] H. K. Scott: Manganese Ores of Brazil. *Journal of the Iron and Steel Institute*, vol. 57, pp. 188–189 (1900).

[4] *Idem*, p. 212.

[5] *Op. cit.*, p. 406.

THE LAFAYETTE DISTRICT

Manganese deposits were discovered in the Lafayette district immediately after the inauguration of mining operations in the Miguel-Burnier district stimulated a search for manganese ores in Minas Geraes, but it was not until the year 1900 that the district became a regular producer, an output of 31,000 tons coming from the Piquery and São Gonçalo Mines in that year. This was increased to nearly 75,000 tons the following year, a production considerably in excess of that of the entire Miguel-Burnier district. In 1902, the Morro da Mina came in as a producer and firmly established the district as the chief manganese-producing district of Brazil.

Lafayette is a station, located at the edge of the town of Queluz, on the Estrada da Ferra Central do Bresil, about 32 km. south of Burnier and 462 km. from Rio de Janeiro. From the fact that the most important producing mines have been located in its vicinity the district has been generally referred to as the Lafayette district. The most important mine at present is the Morro da Mina, owned and operated by a Brazilian company known as the Companhia Morro da Mina, which has increased its production to 700 tons per day. It is located 7 km. northeast of Lafayette. On the same hill, a German company under the name Mineração de Agua Preta has been working the rubble ores to the east of the Morro da Mina ground. This company is producing at the rate of 2,000 tons per month and in the 6 or 7 years it has been operating has produced a total of 200,000 tons. On a small hill to the southeast of the Morro da Mina, the extension of that zone is being developed by a company known as the Companhia Queluz da Mina. The only other producing mine is the Cocuruto which lies about 40 km. southwest of Lafayette and is connected with the railroad at Christiano, a station 23 km. further south, by a 60-cm. gage line 40 km. in length. This mine has been operated about 7 years and is producing 3,000 tons monthly. It is owned by a Belgian company, the Société Anonyme de Manganese de Ouro Preto, which formerly worked the São Gonçalo Mine. After each had produced about 250,000 tons of ore, the São Gonçalo and Piquery mines were abandoned 10 years ago as worked out. They were located about 15 km. northwest of Lafayette. The Piquery Mine is of particular interest in that it is the only one of which there is is a geological description and it was there that Derby obtained the first evidence of the original character of the manganese rock from which the ores were derived.

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[6] *Op. cit.*, p. 791.

In addition to those mentioned, a number of other deposits have been discovered and prospected to some extent, but apparently the results were not favorable enough to warrant further development as none of them became important producers. It is certain, however, that the district has not been thoroughly prospected and there is every reason to expect that systematic exploration would discover deposits equal to those that have been found.

Geology of the Lafayette District

The geology of the Lafayette district differs markedly from that of the Miguel-Burnier. It lies to the south of the area underlain by the great iron-bearing series, and its ores are found in the basement complex of supposed Archean age which underlies a large part of the state of Minas Geraes. The rocks making up this basement complex are chiefly granite and gneiss with which are associated amphibolite, and micaceous and quartzose schists. There are also small intrusions of diorite and gabbro, mostly in the form of dikes. The granite seems to be intrusive into the gneiss and schist, but the relations between the schist and gneiss are not clear.

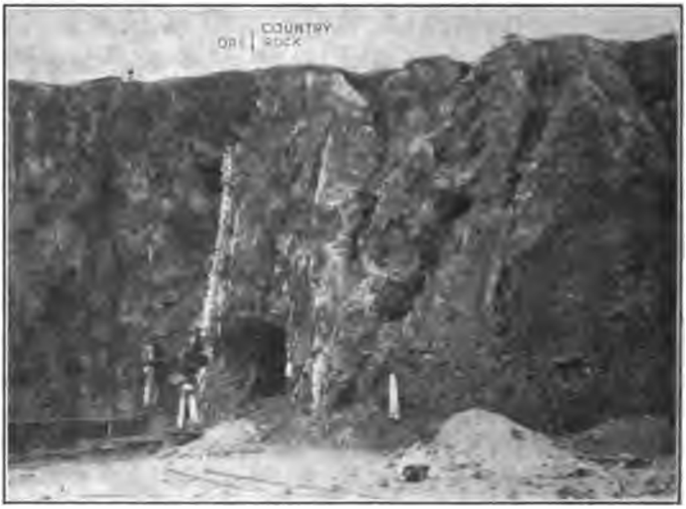

FIG. 1.—SHARP CONTACT OF MANGANESE ORE AND COUNTRY ROCK IN MORRO DA MINA MINE.

The manganese deposits occur as elongated masses of more or less lenticular shape within the rocks of the basement complex. As will be explained below, they represent residual products of decomposition of an original manganiferous rock made up of manganese carbonate and silicates. The immediate wall-rock of the deposits has likewise undergone decomposition, in many instances being nothing more than a clay in which the original rock texture is poorly preserved, so that it is usually difficult to determine its original character. In most cases, however, it seems to have been either gneiss or schist. The contact of wall-rock and ore generally appears quite sharp, as is shown in Fig. 1, but closer examination often reveals small nests and stringers of manganese oxide in the decomposed rock. There are also horses of equally decomposed rock and of the same character within the orebodies themselves.

The Piquery Mine

The geological relations of the Piquery orebody have been described by O. A. Derby in two papers published in 1901 and 1908,[7] and the following account is abstracted from them.

> "The Piquery orebody presents the appearance of a mass of secondary material, or gossan, resulting from the alteration of a vertical dike or vein, some 10 or 12 m. wide. . . . The ore is a hard spongy black oxide, apparently consisting for the most part of psilomelane but with an admixture of other oxides that frequently occur in beautiful crystallizations in the spongy cavities. . . . In the midst of the merchantable ore occur inconstant bands and patches of hard siliceous material with the appearance of a quartzite, but which on examination proves to be composed almost exclusively of a finely granular mass of ashy white manganese garnet. A complete series of alteration phases between perfectly typical garnet rock and merchantable ore can be readily selected, and there can be no doubt that the latter results from the decay and leaching of the former."

Derby describes three phases of the garnet rock that he observed in 1901:

> "1. A very fine-grained, compact and finely jointed rock of bluish-gray color with partings lined with asbestus. Under the microscope the rock is seen to be composed almost exclusively of closely appressed idiomorphic grains of white garnet showing a clear border but with the center highly charged with a fine black opaque powder that appears to be graphite. . . .
>
> "2. A dark brown rock heavily charged with manganese oxide and too friable to permit the preparation of microscopic sections is evidently of the same type but more completely decomposed. . . .

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[7] O. A. Derby: On the Manganese-Ore Deposits of the Queluz (Lafayette) District Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 12, pp. 18–32 (July, 1901).

O. A. Derby: On the Original Type of the Manganese-Ore Deposits of the Queluz District, Minas Geraes, Brazil. *American Journal of Science*, Ser. 4, vol. 25, pp. 213–216 (March, 1908).

> "3. A milky white rock which under the microscope is seen to be composed of about equal parts of garnet and quartz. . . . The quartz in a fine mosaic about the garnet grains and in minute refilled joints is almost certainly secondary."

Residues of manganese garnet were likewise found by Derby at the São Gonçalo, Morro da Mina, Agua Limpa and Barroso Mines. The last two mines were located about 10 km. southeast and 9 km. south of Lafayette respectively. As the result of his observations at these mines, Derby drew the following conclusions concerning the origin of the deposits:

> "The orebodies of the Queluz district are residual deposits derived through decomposition and leaching from an original type or types of rock in which manganese garnet was the most constant and characteristic silicate element. . . . This type, which may appropriately be denominated *queluzite*, is more or less intimately associated at São Gonçalo, Morro da Mina and Barroso with decomposed schistose rocks that evidently contained an original manganese-bearing silicate and which from the absence of recognizable clastic elements and from other characteristics, so far as they can be made out, is presumed to have been an amphibolic schist representing a sheared basic eruptive. . . . In the Agua Limpa schist moreover, the manganese-bearing element is spessartine, as in the orebodies, thus giving greater plausibility to the hypothesis that the relation between these last and the above-mentioned rocks may be a genetic one. If thus related, the orebodies present strong analogies with those of magnetic, titaniferous and chromic iron ores that are now generally considered as magmatic segregations in various types of eruptives, and, all things considered, this hypothesis seems the most plausible one for the manganese ores here discussed."

After the Piquery orebody was worked out and the original manganiferous rock was exposed in the bottom, Derby made a further study of it, the results of which are given in the 1908 paper. He found that the rock consists mainly of "a black, fine-grained, highly jointed and somewhat flaggy rock with the aspect of a limestone, with broad bands and patches of a more massive, yellowish-gray rock with the aspect of a quartzite." The latter is the garnet rock described in the previous paper, but which now turns out to be of secondary importance. The limestone-like rock on treatment with cold weak acid effervesces freely with an abundant separation of gelatinous silica and an insoluble residue containing spessartite and graphite. In places there is also a considerable admixture of rose-colored rhodonite in streaks and patches as the predominant component. Microscopic examination by Dr. Hussak showed the rocks to consist of manganese carbonate, tephroite, and spessartite with a small amount of rhodonite. The paper gives three analyses of these rocks, of which No. I was selected with reference to a supposed high carbonate content, No. II as having a lower carbonate content, and No. III was taken from the earlier paper and represents the garnet rock first found.

For the purpose of comparing the rocks represented by these analyses with the Morro da Mina rocks described below, their mineralogic composition has been calculated approximately on the basis of their described

Analyses of Original Manganese Rock at Piquery

	I	II	III
CO_2	22.62	4.59	
SiO_2	11.80	27.69	38.47
MnO	47.52	57.48	27.90
Al_2O_3	7.50	1.41	21.07
Fe_2O_3	.....	2.48	7.38
CaO	3.76	1.82	4.70
MgO	6.27	4.60	
	99.47	100.05	99.52

mineralogic characteristics. The commercial ores average twice as high in alumina as in ferric iron and on this basis there is just enough ferric iron and alumina in analysis I to take care of the silica in forming spessartite. If all the CO_2 is calculated as rhodochrosite, there is a deficiency of 4 per cent. MnO. But there is more than enough CaO and MgO to take care of this deficiency, so that roughly this rock consisted of 32.6 parts of spessartite and 59 parts of manganese carbonate by weight, or 32 and 68 parts respectively by volume. If in analysis II, the ferric oxide and alumina are calculated as manganese garnet, the CO_2 as rhodochrosite and the remaining silica as tephroite, there is a deficiency of nearly 9 per cent. MnO which is slightly more than covered by the CaO and MgO. On this basis the rock consisted of 12 parts of rhodochrosite, 14.5 parts of spessartite and 75.2 parts of tephroite by weight, or 17, 17.5 and 65.5 parts respectively by volume. Analysis III represents a garnet rock with a little more alumina and about 4 per cent. more silica than is needed to take care of the MnO, CaO, MgO and iron considered as ferrous iron. On this basis it consisted of 65 parts of spessartite, 12.6 of grossularite, and 15.3 of almandite by weight. Actually, of course, the rock consisted of a garnet intermediate in composition between the three but more nearly approaching the composition of spessartite. These calculations show the same great variation in the mineralogic composition of the Piquery rock that is found at the Morro da Mina Mine, except that rhodonite does not appear to have the prominence it attains at the latter.

These new observations necessitated different conclusions and Derby in this second paper says of the deposits that they "seem to be due to the alteration of an original rock with predominant carbonate of manganese and tephroite rather than of spessartite and rhodonite as hitherto supposed." In regard to the genesis of the rock he is silent in this paper, nor does he apply the term *queluzite* to it.

The Morro da Mina Mine

The Morro da Mina Mine is located on a hill 2 or 3 miles north of the town of Queluz, which has an elevation of 1,110 m. above sea level and rises to a height of 200 m. above the surrounding country. A branch line of the railroad connects the mine with the Estrada da Ferra Central do Bresil at Lafayette, so that the ore can be loaded into the cars at the mine and requires no further handling until transferred to ships for exportation at Rio de Janeiro. The mine presents one of the most remarkable manganese deposits in the world both in respect to size and quality

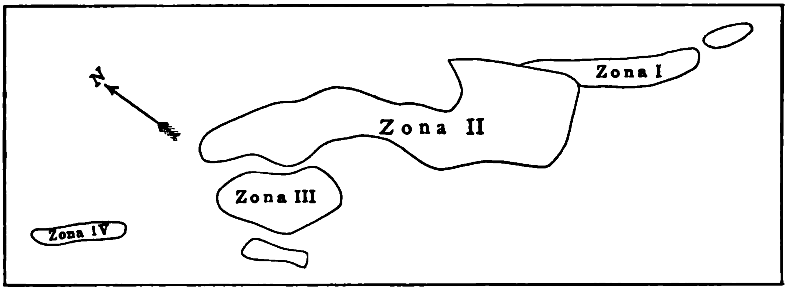

Fig. 2.—Horizontal Section of Morro da Mina Orebodies at Level 55 M. below Summit of Hill.

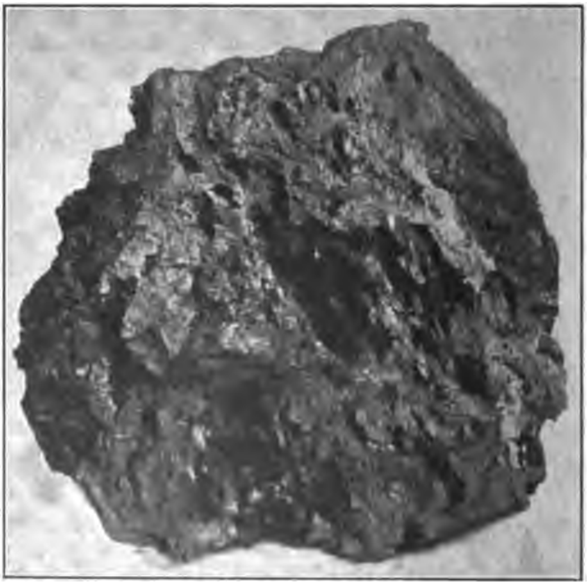

Fig. 3.—Somewhat Drusy Massive Psilomelane, the Most Common Type of Ore at the Morro da Mina Mine.

of the ore; in fact, the manager of the mine, J. de A. Lustosa, says it is the largest known deposit of high-grade manganese ore. Since it became an important producer in 1902, it has yielded a total of over 1,000,000 tons, and in 1915 its production was about 200,000 tons. Development work has proved an ore reserve of 10,000,000 tons.

The orebodies occur at the top and on the flanks of the hill as a series of more or less overlapping lenses extending in a direction N 35° W., with a vertical dip and a pitch of 45° to the southeast. The relative positions, shapes and sizes of the orebodies that have been developed as they occur on the level 55 m. below the top of the hill are shown on the mine map in Fig. 2. The four largest are known as Zona I, II, III and IV respectively and have maximum dimensions of 200 by 30 m., 420 by 120 m., 140 by 70 m., and 100 by 20 m. In depth Zona I and Zona II have been cut by a development tunnel 130 m. below the summit of the hill, indicating that the ores extend at least that far down. In addition to the ores *in situ*, a large part of the hillside below the ore outcrops is covered with rubble ore derived from them.

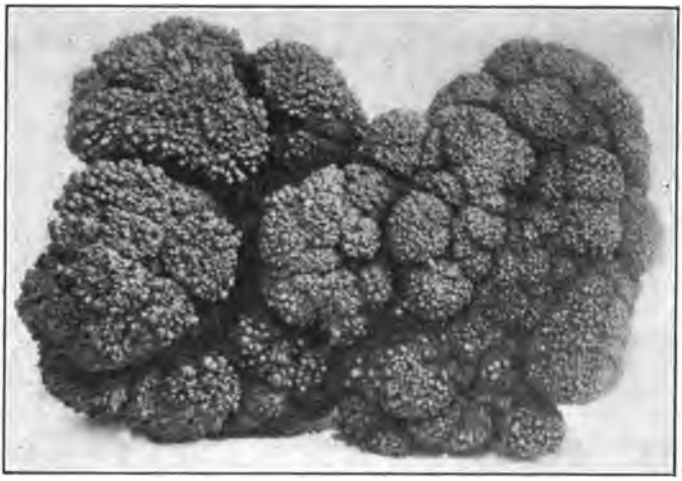

Fig. 4.—Cauliflower-like Cluster of Manganese Oxides Found Quite Frequently in the Morro da Mina Ores.

The ore consists for the most part of psilomelane, which occurs in a variety of forms. Most commonly it is simply more or less drusy massive psilomelane as in Fig. 3, but mammillary, botryoidal and concretionary forms are abundant and frequently quite elaborate. The surface of some of these, except for the black color, reminds one of cauliflower. A typical specimen is illustrated by Fig. 4. Associated with the psilomelane is considerable manganite and pyrolusite which occur for the most part as cavity linings and fillings in the former. The manganite occurs lining the cavities both in radiating groups of acicular crystals and in distinct prismatic crystals. The pyrolusite is often pseudomorphic after manganite showing that it has in part been derived from it. The average composition of the ore as shipped is:

Average Analysis of Morro da Mina Ore

Water at 100° C.	2.50 per cent.
Volatile	12.40 chiefly oxygen.
Insol. Residue	3.46
Fe_2O_3–Al_2O_3	8.75 alumina about twice ferric content.
Silica	1.76
P	0.069
S	Absent
Manganese	50.47

It is of interest to compare this average with the following mean of analyses of cargoes of manganese ores landed at Middlesborough, England, during the years 1897 to 1906 as compiled by L. L. Fermor[8] for samples dried at 100° C.:

	India	Russia
Mn	50.86	49.58
Iron	6.31	0.83
Silica	5.71	10.17
P	0.127	0.161
Alumina, etc.	6.80	12.77

These figures indicate that the three countries produce very high-grade ore with the advantage in favor of the Brazilian ores, particularly over the Russian, as regards silica and phosphorus.

The ore is mined by hand for the most part in open cuts. A number of tunnels and adits have been driven, but these were intended either for development purposes or to connect the various open cuts to give access to the loading platforms and bins at the railroad. The method in vogue is to strip the orebody of such overburden as it may carry, and then as the ore is mined screen it over iron screens with 0.8-in. square openings. The oversize is the merchantable ore. The screenings constituting 15 per cent. of the crude ore, carry 34 to 35 per cent. Mn and are being stored apart from the waste to be beneficiated at some future time. The ore *in situ* furnishes two-thirds of the present output, and one-third is derived from workings in the rubble ores.

THE ORIGINAL MANGANESE ROCK

Though decomposition of the original manganese rock has extended to considerable depth in places in the Morro da Mina Mine, to at least 130 m. as demonstrated in the exploratory tunnel previously mentioned, portions of the rock have escaped alteration and are well exposed in the mine workings. A large mass of the rock is actually exposed at the surface at the southwest corner of the outcrop of Zona II. It is characterized by a predominance of the silicates, and especially garnet, and for that reason has not succumbed to the processes of weathering. The abrupt transition from this rock to ore is shown in Fig. 5, and it is only the presence of numerous stringers, tongues and patches of manganese oxide penetrating the rock close to the ore that makes clear their genetic relations. More interesting and instructive exposures are found in the 130-m. tunnel. This tunnel crosscuts in a westerly direction the decomposed country rock for 120 m. to Zona I, then cuts diagonally across it in good ore for 150 m., and on emerging penetrates 50 m. of the manganese rock to the ore of Zona II. After again cutting 50 m. of ore, a small horse of the manganese rock is encountered with ore on the opposite side. The four contacts of ore and rock exposed in this tunnel showed the same rapid change from one to the other with only a narrow transition zone marked by stringers and tongues of the black manganese oxide penetrating the manganese rock.

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[8] L. L. Fermor: Manganese-Ore Deposits of India. *Memoirs of the Geological Survey of India*, vol. 37, Pt. III, p. 518 (1909).

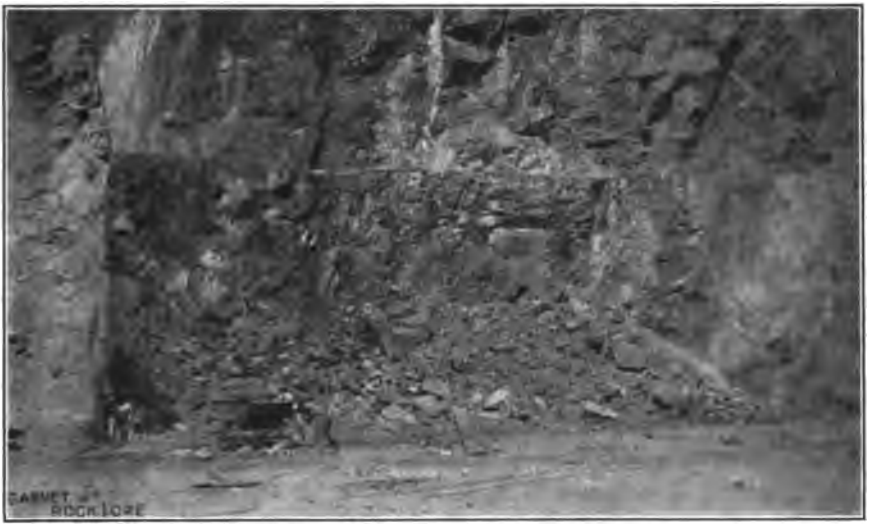

FIG. 5.—SHARP CONTACT BETWEEN THE MANGANESE ORE AND THE ORIGINAL MANGANESE ROCK CHARACTERIZED BY AN ABUNDANCE OF MANGANESE GARNET.

Most of this rock has the appearance of a fine-grained dark gray crystalline limestone and is easily scratched with the point of the pick. Its specific gravity, however, is considerably above that of limestone. Here and there are brown patches and streaks with a violet tinge that consist of massive garnet, and there are frequently spots and stringers of pink rhodonite that at once attract attention. A closer examination reveals the presence of sufficient light pink silicate in much of the material to give a pinkish tone to its dominant gray color.

Examination of thin sections shows that the essential constituents are manganese carbonate, spessartite, rhodonite and tephroite. Taking the average of all the rock, the manganese carbonate is the most abundant mineral and the tephroite the least abundant. The spessartite is more widespread in its distribution but probably not much in excess of the rhodonite in actual quantity. The relative quantity of the different minerals varies most widely, so that some of the rock consists almost entirely of one of the minerals while some has them present in almost equal quantity, with the exception of the tephroite, which, in the sections examined, was never present in more than subordinate amount. In this respect the rock differs from the Piquery rock described by Derby in which tephroite was prominent and rhodonite a subordinate constituent. Which of the two minerals was formed in a given case depended undoubtedly on the relative amount of silica available, a low silica content giving rise to the formation of the orthosilicate tephroite and a higher silica content to the metasilicate rhodonite.

A thin section of a specimen most closely resembling crystalline limestone consists almost entirely of anhedral grains of manganese carbonate averaging between 0.16 and 0.25 mm. in diameter. That the carbonate is essentially manganese carbonate rather than calcium or magnesium carbonate is shown by the specific gravity of the rock, which is 3.55. The manganese carbonate is swarming with flakes of graphite ranging in size from 0.03 to 0.08 mm. and somewhat less abundantly with small garnets ranging from 0.01 to 0.03 mm. in diameter. In the entire slide there are only a few small areas containing rhodonite or tephroite. The minute garnets in this slide differ from the larger garnets in the more siliceous phases of the rock in being perfectly clear, whereas the latter are characterized by the presence of numerous minute black inclusions such as described by Derby as graphite in the rocks studied by him. Another specimen of more siliceous looking rock with a pinkish tone and streaks of rhodonite shows in thin section manganese carbonate, garnet, rhodonite and tephroite in abundance. The constituents average in size from 0.08 to 0.16 mm. with a few larger crystals of rhodonite. As is generally the case, the garnets are pronouncedly euhedral while the other minerals are anhedral. A specimen of particularly siliceous looking rock having a specific gravity of 3.8 consists for the most part of rhodonite with which is associated considerable euhedral garnet. The rock itself is light brown in color with a distinct pinkish tone. In addition to the silicates there is also an appreciable amount of the carbonate present. The garnets range in size from 0.10 to 0.15 mm. and the rhodonite crystals average about two or three times as large. Other thin sections show essentially similar features.

The manner in which this rock undergoes alteration to manganese oxides is very interesting. A specimen undergoing oxidation is illustrated in Fig. 6. As the oxidizing solutions penetrate the rock they first break down the manganese carbonate, tephroite and rhodonite and leave behind embedded in the manganese oxides the grains of garnet. That is, oxidation does not run ahead decomposing the manganese carbonate first

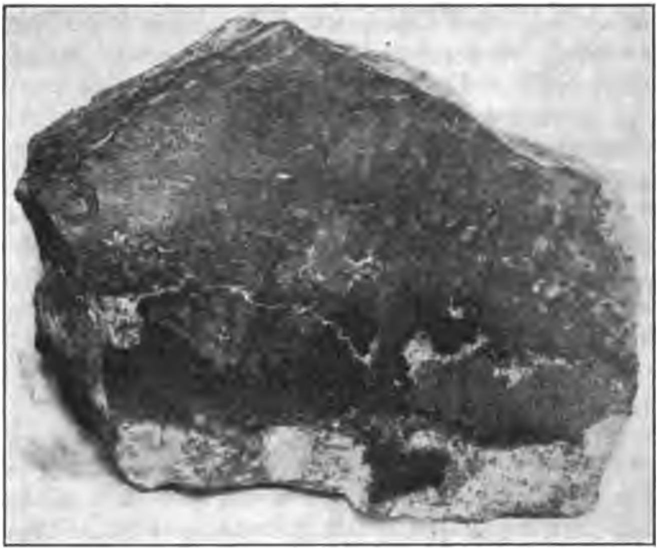

FIG. 6.—POLISHED SURFACE OF THE ORIGINAL MANGANESE ROCK IN PART ALTERED TO OXIDES OF MANGANESE. (Black areas and patches represent the manganese oxides.)

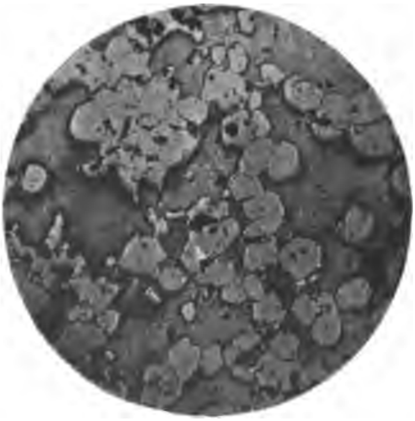

FIG. 7.—RESIDUAL GARNET GRAINS IN A GROUNDMASS OF MANGANESE ORE RESULTING FROM THE OXIDATION OF THE MANGANESE ROCK.

and then as a more advanced stage tephroite and rhodonite successively, as one might expect; but the three minerals are replaced simultaneously at the very front of the advance of oxidation. A magnified polished surface of rock which has undergone this first stage of alteration is shown in Fig. 7, in which the garnet crystals are seen disseminated in a matrix of manganese oxides. Such material from this mine was described by Derby in his first paper and both minerals considered primary elements of the rock. In his second paper he doubted this first interpretation and was more inclined to regard the manganese oxide a residue of the alteration of original carbonate and silicate minerals with retention of the spessartite. The evidence of the thin sections makes clear that the latter is the true explanation. As this material is further worked on by the oxidizing solutions, the garnet begins to succumb around the periphery of the grains and along fractures until finally it too has been replaced by the manganese oxides.

GENESIS OF THE LAFAYETTE TYPE OF DEPOSIT

The manganese deposits of the Lafayette district inevitably call to one's mind the Indian deposits which likewise consist of manganese oxides associated with manganese silicate rocks, and as we shall have occasion to refer to them in the discussion of their genesis, it will be helpful to summarize at the outset certain of the salient features of the Indian deposits as described by L. L. Fermor.[9] There are two main types of manganese deposits in India, associated respectively with the *kodurite* and *gondite* rocks. The former are found in the Vizagapatam district on the east coast, the latter in the Central Provinces and other parts of India. Typical kodurite is composed of potash feldspar, manganese garnet and apatite, with or without pyroxene. From the mineralogical and chemical composition of the rock and its geologic relations, Fermor concludes that kodurite is an igneous rock. It was subjected to alteration under oxidizing conditions and gave rise to the manganese ores. He finds that the manganese garnet is the most stable mineral and is often left behind in a matrix of psilomelane. To the gondite rocks he gives an entirely different interpretation. He thinks they were originally deposited as sediments and the manganese which they contain as chemical sediments, most probably in the form of oxide, though perhaps to a limited extent as carbonate. These sediments were later subjected to intense dynamo-metamorphism, and where they consisted of relatively pure chemical sediments of manganese were converted into compact psilomelane and braunite. Where alumina and silica had also been deposited, spessartite and rhodonite were formed, and if there was an excess of silica it was crystallized as quartz. The resultant rock is called gondite. Typical gondite is a very fine-grained rock consisting of tiny round grains of spessartite set in a mosaic of quartz. Where alumina is deficient rhodonite is found. The ores associated with the gondite rocks, Fermor believes are the result of combined decomposition and replacement of them by waters containing CO_2 and O, but that this action took place at considerable depth and that the carbon dioxide was a portion of that liberated in the metamorphism of the rocks of the region and the oxygen a portion of that liberated in the conversion of the original manganiferous sediments into the manganese silicate rocks. A small proportion of softish and more or less porous ore he attributes to the later surficial alteration of the gondite.

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[9] *Op. cit.*

An explanation of the genesis of the Lafayette manganese deposits involves two problems: First, the genesis of the original manganese rock; second, the alteration of it to manganese oxides, that is, to the manganese ores.

As the evidence is beyond question in regard to the second stage in the formation of these ores, that problem will be discussed first. The manner of occurrence of the orebodies, as for instance the change in depth of the Piquery ores into the manganese rock, the abundant evidence of the alteration near the surface of the manganese rock into ore and the microscopic evidence of the same phenomenon, point unmistakably to the derivation of the ores from the manganese rock under conditions of weathering in the zone of oxidation. That is, the ores were formed under conditions similar to those that gave rise to the ores of the kodurite rocks of India and not as Fermor thinks most of the ores in the gondite series were formed. The chemistry of the alteration of the Brazilian manganese rock is simple compared with that of the kodurite. The principal constituents to be removed are the CO_2 of the manganese carbonate and the SiO_2 of the silicates. Examples of the efficacy of meteoric waters for that purpose are so abundant and generally recognized that this part of the process hardly requires elaboration. The universal alteration of iron carbonates to oxides and hydroxides in the zone of oxidation and the enormous amount of silica removed from the iron silicates in the formation of the Lake Superior iron ores are illustrations of the same chemical actions on similar compounds of a chemically closely related element. In the kodurite rocks the removal of considerable alumina was also necessary and the chemistry of this was the only step that Fermor found at all difficult to explain. But even the very slight solubility of alumina in ordinary dilute meteoric waters he thought would suffice for its ultimate removal. This difficulty, however, hardly enters in our problem. Except in the case of the local phases of garnet rock, which has anyhow to a large extent resisted alteration, the alumina content of the original rock is no higher, and the few available analyses would indicate actually a little lower, than that of the ores; and, furthermore, the total amount involved in either is not over 5 per cent. Consequently the problem of the removal of silica does not confront us.

The ores have been described as somewhat porous and drusy, but by no means to such an extent as would be called for if they merely represented the residual product of leaching of manganese carbonate and silicates. Nor is there evidence to indicate that their present more compact form is due to shrinkage in volume or compression of the residual manganese oxides; but the evidence both megascopic and microscopic shows that the decomposing solutions deposited manganese oxide, volume for volume for the silica and carbon dioxide removed. Meteoric waters encountering such large quantities of manganese carbonate certainly took a great deal of it into solution as manganese bicarbonate which would be deposited at other points as the oxide; and in this way the deposition of manganese oxide accompanied the leaching of the other constituents of the rock. The alteration of the manganese rock to ore was, therefore, one of simultaneous leaching and addition, resulting in a relatively compact mass of the oxides. The present drusy character of some of the ore is clearly the result of subsequent action of solution and redeposition by meteoric waters upon the manganese oxides formed in this way.

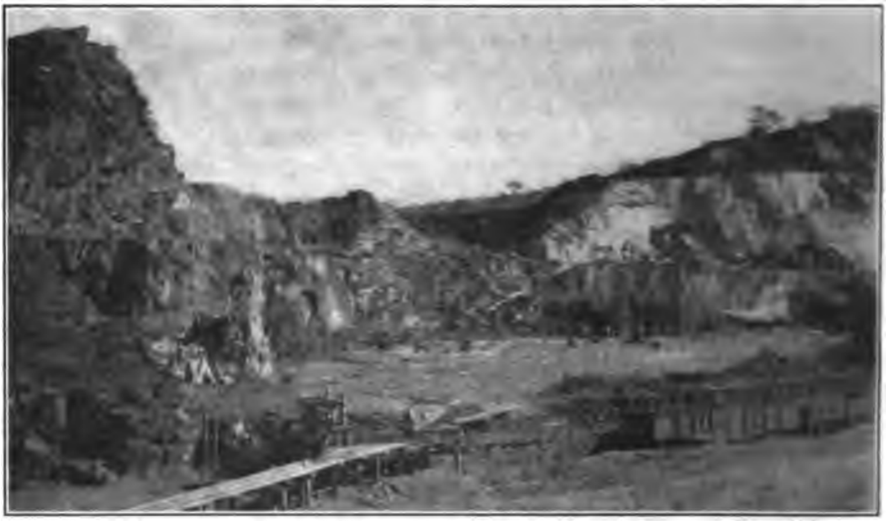

FIG. 8.—VIEW OF SOUTH END OF OPEN CUT OF THE MORRO DA MINA MINE, SAID TO BE THE LARGEST KNOWN DEPOSIT OF HIGH-GRADE MANGANESE ORE.

The problem of the origin of the manganese rock is not so easy to solve. One is at once confronted with the fact that our knowledge is rather meager concerning the exact nature of the rocks with which it is associated and its geologic relations to them, a full knowledge of which is essential to a final solution. The chemical, mineralogic and petrographic characters of the rock, however, do rule out some explanations, and point strongly to another. Further, the geologic relations that are reasonably well established are not at variance with this explanation but in a measure support it. That is, the sum of all evidence points to an analogy of this rock with the gondite of India. We believe that the rock is the product of dynamo-metamorphism of manganese sediments deposited in the form of manganese carbonate with varying but considerable quantities of silica and varying but smaller quantities of alumina. These sediments differed, therefore, initially from the gondite sediments in averaging much lower in silica, and by the deposition of the manganese in the form of carbonate instead of oxide; and this initial difference in composition accounts for the present difference in mineralogy of the two rocks.

The greater resistance to decomposition of the garnet in the manganese rock at the Piquery mine led Derby in 1901 to the erroneous conclusion that it consisted essentially of manganese garnet and to establish the rock type *queluzite* to which he attributed an igneous origin. Later, in 1908, he found that the garnet rock was but a subordinate phase of an entirely different type of rock which is essentially the same as that which we found abundantly at the Morro da Mina Mine. Attention has been called to the fact that in this later paper he does not apply the term queluzite to this rock and ventures no opinion as to its genesis. Though he did not specifically retract his earlier statement, the presumption is that, in the light of the discovery that the original rock was entirely different from what he had supposed, he abandoned his interpretation of its origin. In the meantime, L. L. Fermor,[10] on the basis of Derby's first paper, drew an analogy between this rock and his kodurite as regards their genesis, though calling attention to their dissimilarity in composition. A persistence of this same view is manifested in the statement by Harder and Chamberlin[11] in their discussion of these ores that "manganese ores associated with igneous rocks, such as those described above, occur abundantly in India—" and the statement by Harder[12] that "The relation of the manganese rock to the enclosing crystalline rocks has not been definitely determined; it may be interlayered with the gneiss or crystalline schist, or perhaps intrusive into them." The chemical and mineralogic composition of this rock as now established at both Piquery and Morro da Mina effectually precludes an igneous origin for it and demonstrates the presence of an original carbonate rock.

Even before Derby's second paper appeared, a view at variance with it was expressed by his associate E. Hussak.[13] He regarded the spessartite rock described by Derby together with a banded spessartite-rhodonite rock, which he mentions as occurring on the periphery of the Piquery orebody, the products of contact metamorphism by the eruptive gneiss of impure manganese carbonate sediments. The postulation of the derivation of the silicates from manganese carbonate in advance of the discovery of the presence of large amounts of carbonate is interesting. Beyschlag, Krusch and Vogt[14] are inclined to carry Hussak's ideas a step further and suggest that the manganese may have been contributed as part of the process of contact metamorphism. That this was not the case is proved by the fact that the carbonate still present, constituting a large percentage of the rock, is the manganese carbonate; and that consequently the original carbonate was a manganese carbonate. There remains to decide between the view of Hussak that the rock is the result of contact metamorphism of sedimentary beds of impure manganese carbonate and our own that it is a product of dynamo-metamorphism of such beds.

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[10] *Op. cit.*, pp. 273–274.
[11] *Loc. cit.*, p. 405.
[12] *Loc. cit.*, p. 790.

In the first place, the country rock of the ore deposits is not everywhere gneiss, but is in some cases schist, and these schists are admitted to be metamorphosed sediments. Nor are all of the gneisses necessarily igneous, though it has been generally held that they are such for the most part. Consequently the presumption is against assuming an igneous contact for all of the deposits. In the second place, the texture and appearance of the silicates is not that of typical skarn minerals, or products of contact metamorphism; but it is that of silicates crystallized in a carbonate rock that has been subjected to regional metamorphism. The reply might be made that the whole region has been subjected to dynamo-metamorphism subsequent to the contact metamorphism, but it is equally true that the texture of the rock is not what one would expect to result from a skarn rock subjected to regional metamorphism. On the other hand, we cannot escape the conclusion, demanded by both hypotheses, that there first existed impure manganese carbonate, and it is obvious that the rocks have undergone regional metamorphism. The inevitable result would be a rock such as we have. Our explanation is, therefore, in harmony with the available evidence and makes the manganese rock of the Lafayette district genetically identical with the gondite of India, the mineralogic difference of the two rocks being due to their initial difference in chemical composition.

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[13] Eugen Hussak: Ueber Atopit aus den Manganerzgruben von Miguel Burnier, Minas Geraes, Brasilien. *Centralblatt für Mineralogie, Geologie und Paläontologie*, 1905, pp. 240–245.

Eugen Hussak: "Über die Manganerzlager Brasiliens. *Zeitschrift für praktische Geologie*, vol. 14, pp. 237–239 (1906).

[14] Beyschlag, Krusch, Vogt: *Die Lagerstätten der nutzbaren Mineralien und Gesteine*, vol. 2, p. 596 (1913).

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Recent Geologic Developments on the Mesabi Iron Range, Minnesota

BY J. F. WOLFF, E. M.*, DULUTH, MINN.

(New York Meeting, February, 1917)

During the past 4 or 5 years, much has been added to the detailed geologic knowledge of the Mesabi Range. This has not been in the direction of discovery of any new fundamental facts, but of detailed study, subdivision and correlation of different parts of the whole formation and of individual orebodies. Prior to this time, mining engineers in the district were so engaged with the commercial interests of exploring and developing orebodies that close geologic study and subdivision was done in only a few instances. The demand for refinements of work in this direction has caused extensive structural subdivision and correlation to be done in all exploration work during the past 4 years by the engineers of the Oliver Iron Mining Co. Such work has become a commercial necessity rather than a scientific refinement, and at the present time is being extended to all parts of the Range.

In the summer of 1914, the author of this paper wrote a series of articles on the orebodies and special features of the Mesabi Range, which was published in the *Engineering and Mining Journal*, issues of July 17 to Aug. 7, 1915. Since that time studies of subdivision of the iron-formation have been extended considerably. The principle feature of this paper is the presentation of the subdivisions of the iron-formation. To this is added a discussion of the relations of the orebodies to the gentle folding and fracturing of the formation, and special features of the Range.

The outline will be as follows:

I. General geology.

II. Subdivision of Biwabik iron-formation.

III. Relation of orebodies to folding and fracturing of the iron-formation.

IV. Special features.

Acknowledgment is due to W. J. Olcott, President, and John Uno Sebenius, General Mining Engineer, of the Oliver Iron Mining Co., for permission to use the information presented in this paper.

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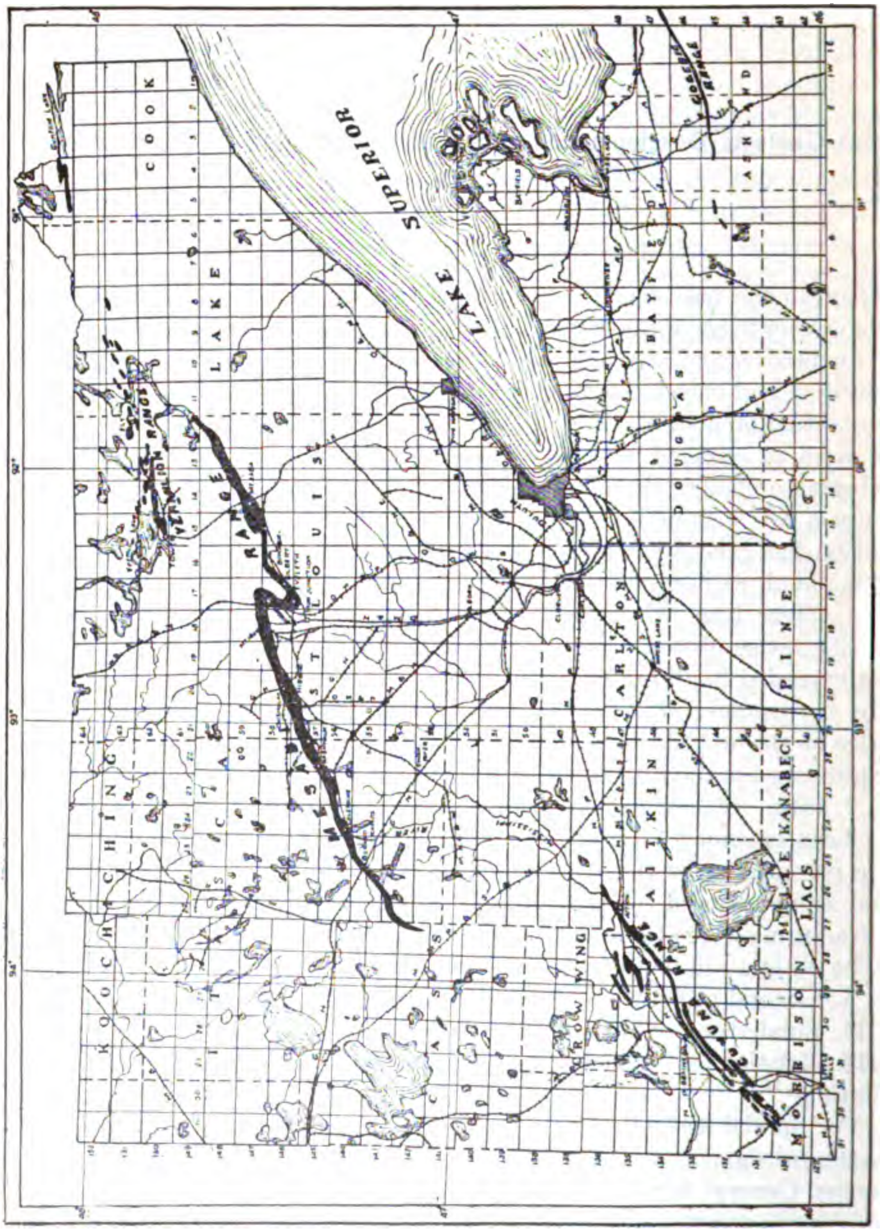
* Mining Engineer, Oliver Iron Mining Co., Duluth, Minn.


LAKE SUPERIOR
COOK
LAKE
ST. LOUIS
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KOOCHICHING
CASS
CROW WING
AITKIN
CARLTON
PINE
KANABEC
MILLE LACS
MORRISON
DOUGLAS
BAYFIELD
ASHLAND
MESABI RANGE
VERMILION IRON RANGE
GOGEBIC RANGE
CUYUNA RANGE
92°
93°
94°
91°

FIG. 1.

General Geology

Correlation

Although the general geology of the Mesabi Range has been described most exhaustively in *U. S. Geological Survey Monographs* Nos. 43 and 52, and elsewhere, a short synopsis may be a necessary preliminary to the detailed discussion of the Range presented in this paper, because many of the readers of this article may be unfamiliar with the above publications or the general geology of the Mesabi district.

The topographic feature of the Mesabi Range is a line of fairly prominent and continuous hills ranging in elevation from 1,400 to 1,900 ft. above sea level, composed of a complex of granites, greenstones, green schists, slates, graywacke and conglomerate. Resting unconformably upon this basement complex, and sloping away at a gentle angle to the southeast, is a series of sedimentary rocks, the middle member of which is iron-bearing. The outcrop of this iron-bearing member is the geologic feature known as the Mesabi (or Missabe) Iron Range. Within this formation the iron ore-bodies are found. Its outcrop has been traced by explorations from Sec. 12, T. 142 N., R. 25 W., northeastward to Birch Lake, in Sec. 26, T. 61 N., R. 12 W., a distance along the strike of about 112 miles. Its width varies from 3/8 to 3 miles, due to variation in the dip and thickness. The sedimentary series above mentioned consists of a basal quartzite, named Pokegama, an intermediate iron-formation, named Biwabik, and a top black slate, named Virginia slate.

The location and general extent of the Range is shown on the Map, Fig. 1. The general relations of the iron-bearing member to associated rocks are shown by the cross-section, Fig. 2, better than further description could explain.

Instead of the complete correlation table as given by the U. S. Geological Survey, a simplified geologic column is herewith given, which is adequate for the engineer in the district.

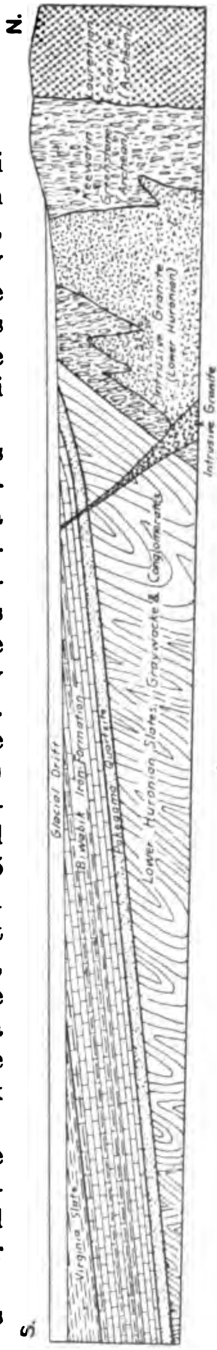

N.
Laurentian Granite (Archean)
Keewatin Greenstone (Archean)
Intrusive Granite (Lower Huronian)
Intrusive Granite (Keweenawan)
Lower Huronian Slates, Graywacke & Conglomerate
Glacial Drift
Biwabik Iron Formation
Pokegama Quartzite
Virginia Slate
S

Fig. 2.—Generalized Cross-Section Showing Relation of Iron Formation to Associated Rocks.

Quaternary System		
Pleistocene Series	Glacial drift, 0 to 300 ft.	
Unconformity.		
Cretaceous System	Conglomerates and shales, 0 to 50 ft.	
Unconformity		
Algonkian System Keweenawan Series	Duluth Gabbro Embarrass Granite	East end of range.
Unconformity Upper Huronian Series (Sedimentary)	Virginia slate 0 ft. to great thickness. Biwabik Iron-formation, 475 to 775 ft. Pokegama Quartzite, 50 to 150 ft.	
Unconformity Algonkian and Archean Series	Basement complex of Slate-graywacke-conglomerate series, granites, greenstones, greenschists and porphyries.	

From a commercial standpoint the Upper Huronian series, Virginia Slate, Biwabik Iron-formation, Pokegama Quartzite are the important rocks of the district.

The characteristics of the different formations need not be discussed. This will be done for the iron-formation later on.

Orebodies

The Upper Huronian Quartzite-Iron-formation-Slate series is an ordinary (except for the iron) series of clastic sediments, deposited in fairly shallow water, contemporaneous with the middle member of which was deposited or precipitated out of solution an enormous quantity of iron, chiefly in the form of a ferrous silicate, $FeSiO_3$, occurring as coarse and minute green granules of the mineral greenalite, embedded in a matrix of chert. Iron carbonate, grunerite, amphibole, actinolite, and hematite occur in small quantities. Original magnetite in bands and finely disseminated grains occurs in much larger quantity than has been recognized heretofore, in all parts of the district. The bedding-planes of all three members are approximately parallel. Interbedded with the iron-formation in certain horizons are numerous slate layers, varying in thickness from a few inches to many feet. The iron-formation from Virginia slate to quartzite varies in thickness from 475 to 775 ft., the average being about 600 ft.

After this series of sediments had been laid down, earth movements raised them above water level, allowing erosive agencies to cut through the overlying slate into the underlying formations. These earth movements warped the formations and cracked the brittle iron-formation quite extensively, allowing surface waters to enter its upturned edges. Especially where such cracking was pronounced, the ground-waters entering the iron-formation, carrying carbon dioxide in solution, attacked the ferrous iron compounds and oxidized them. In such localities much of the ferrous silicate has been changed to a hematite-chert rock, the hematite occurring as bands and as disseminated particles in the chert, the rock still retaining its solidity. The whole iron-formation, whether thus altered or not, is a ferruginous chert with interbedded slate layers and locally is called "taconite." This term should apply strictly to the ferruginous chert and not to the slate layers, though in some horizons the slate bands and chert layers are so intimately interbedded as to make a distinction quite impossible.

Where the cracking has been most intense the circulation of groundwater has been most vigorous, the solvents in the ground-waters have leached out the silica and other minor constituents of the rocks and have oxidized and left in place the iron. Such residual material now constitutes the orebodies. They are surrounded on all sides by the rock walls of the iron-formation from which they are derived. Pore-space was developed by this removal of silica and the settling or slumping in place of the layers of the orebodies is a characteristic feature. The typical orebody thus developed has a trough structure and an irregular trough shape. Orebodies vary in size from a few acres to several hundred acres and from a few feet to several hundred feet in thickness. In many places small troughs unite to form a large one. Fig. 3 shows the shape and area of a typical orebody. While the trough orebody is the typical one, there are two other types, namely, the flat-layered body and the fissure-type orebody. The former is either the remnant left by the erosion of a former trough-body or it is an ore layer continuing down the dip from a trough-body. Usually such a layer has a rock (slate) capping. The fissure-type orebody is an incompletely developed trough-body and is usually associated with a larger trough orebody. Fig. 4 shows the development of a trough orebody on the axis of an anticline, where cracking has been pronounced. All four stages here shown can be observed in the field. Notice the slumping of ore layers near the rock walls. The features of the three different types of orebodies were discussed at length in the *Engineering and Mining Journal*, July 17 to Aug. 7, 1915, and will not be repeated here.

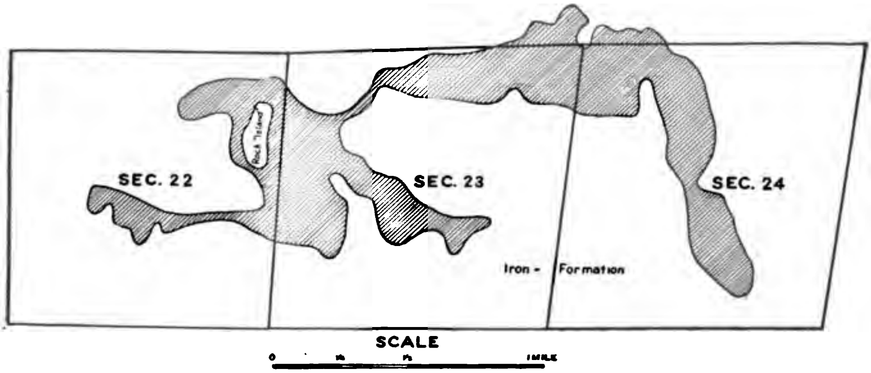
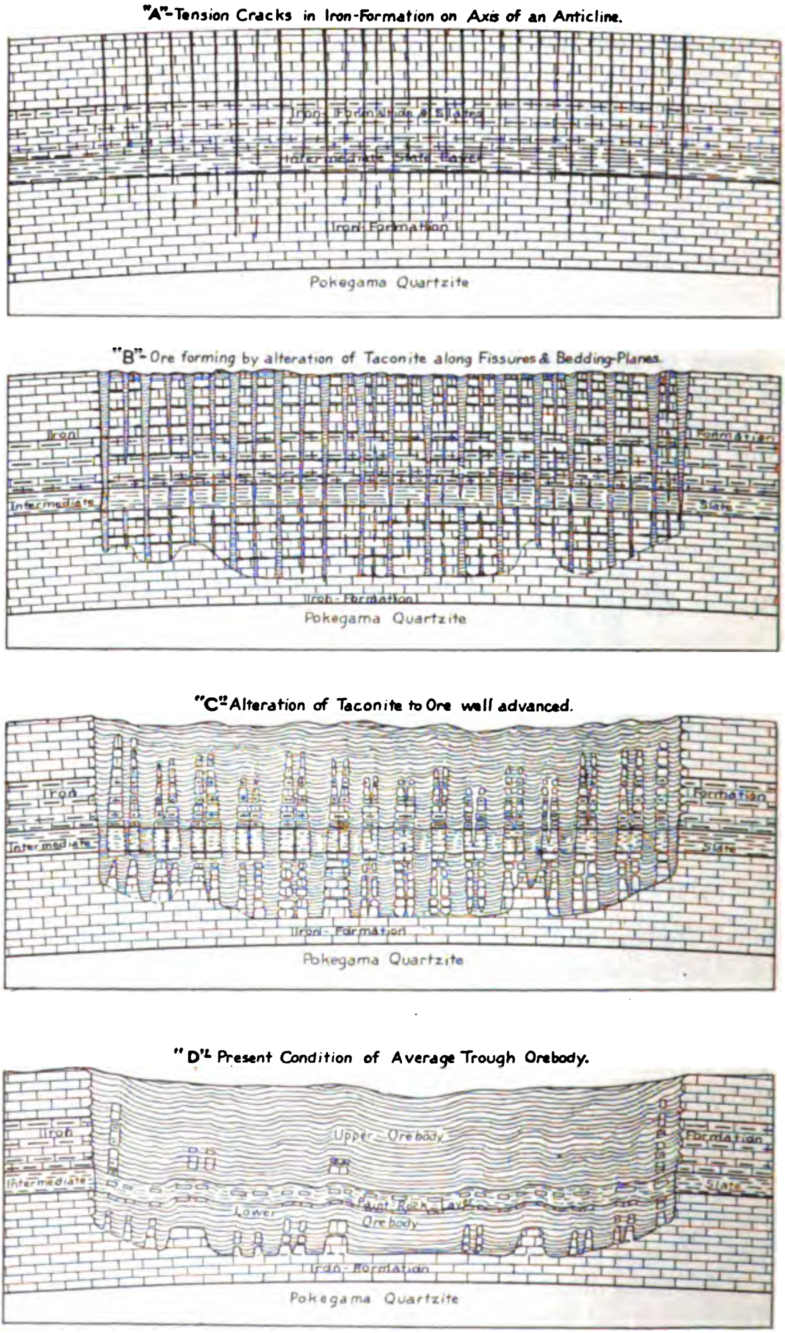


FIG. 3.—PLAT SHOWING SHAPE AND AREA OF TYPICAL OREBODIES.



"A"-Tension Cracks in Iron-Formation on Axis of an Anticline.

"B"-Ore forming by alteration of Taconite along Fissures & Bedding-Planes.

"C"-Alteration of Taconite to Ore well advanced.

"D"-Present Condition of Average Trough Orebody.

FIG. 4.—CROSS-SECTIONS SHOWING STAGES IN DEVELOPMENT OF A TROUGH ORE-BODY ON THE AXIS OF A GENTLE ANTICLINE.

Covering the entire iron-formation, except at a few isolated spots, is a mantle of glacial drift, varying in thickness from a few feet to 300 ft.

SUBDIVISIONS OF THE IRON-FORMATION

Detailed Subdivision

The item of principal interest and most practical value to mining men in the Mesabi district, presented in this paper, is the subdivision of the iron-formation into its several horizons. As indicated above, the engineers of the Oliver Iron Mining Co. have found it absolutely essential to do this kind of work in the exploration and development of their orebodies. Undoubtedly others will meet with the same experience. The determination of the structure of an orebody depends upon the proper subdivision and correlation of the different ore and rock layers in the drill records.

Such correlation, not only in one area but over the length of the Range, has shown that the formation can be subdivided into four principal horizons, each of which has remarkably uniform characteristics from one end of the Range to the other. Each horizon can be subdivided further. Of course, the thickness of each horizon is not uniform because the thickness of the whole formation varies. There are local peculiarities in some places, but these do not destroy the major subdivisions. This subdivision and correlation has been done chiefly by F. B. Cronk, Mining Engineer for the Oliver Iron Mining Co., and the writer.

Fig. 5 shows the four subdivisions which can be made in any part of the Mesabi district. From the top down they are, Upper Slaty Horizon, Upper Cherty Horizon, Lower Slaty Horizon and Lower Cherty Horizon. They are named from the predominant physical characteristic of the rock in them. Commercially the two cherty horizons and Lower Slaty Horizon are most important, as the principal orebodies occur in them. The two cherty horizons contain very few slate seams, but the slaty horizons contain a considerable number of interbedded chert layers and a great amount of greenalite. The lower

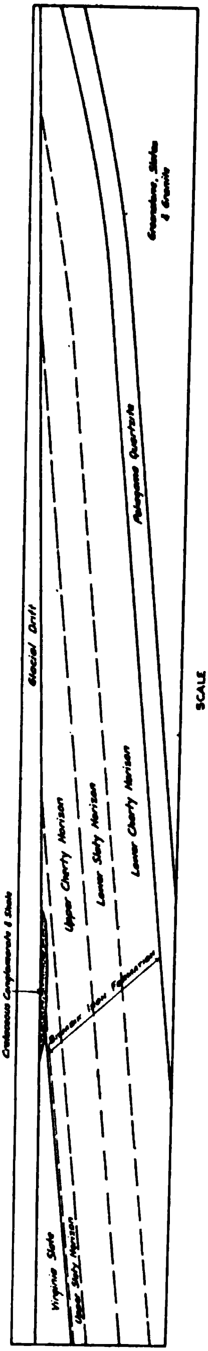


FIG. 5.—GENERALIZED CROSS-SECTION SHOWING SUBDIVISION OF THE BIWABIK IRON-FORMATION.

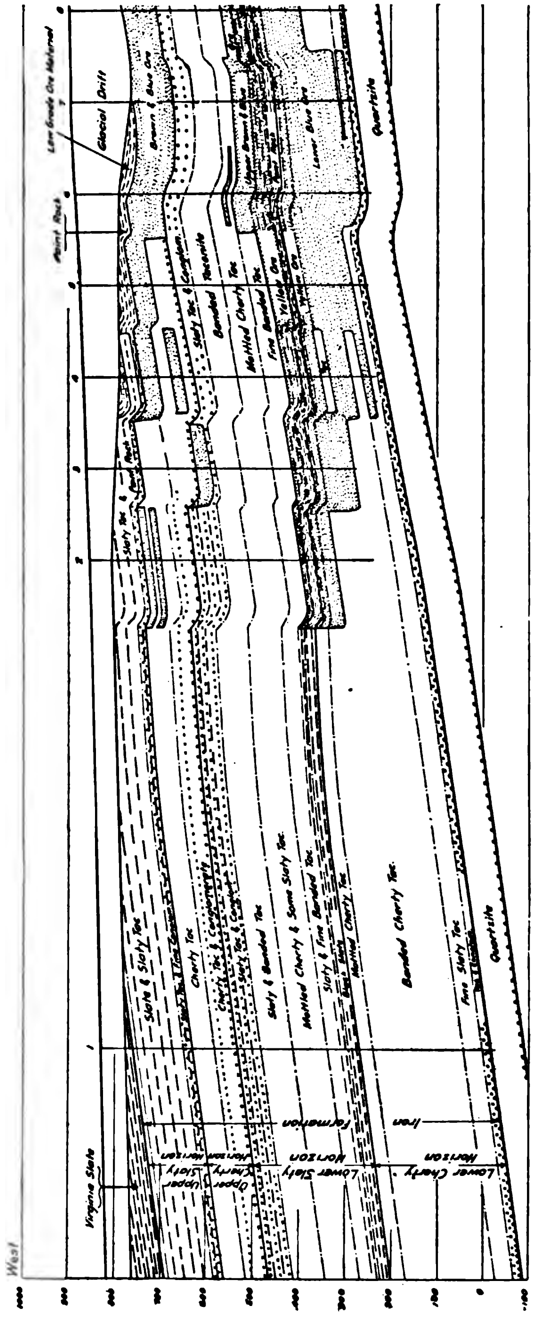


FIG. 6.—LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION. (*Continued on two succeeding pages.*)

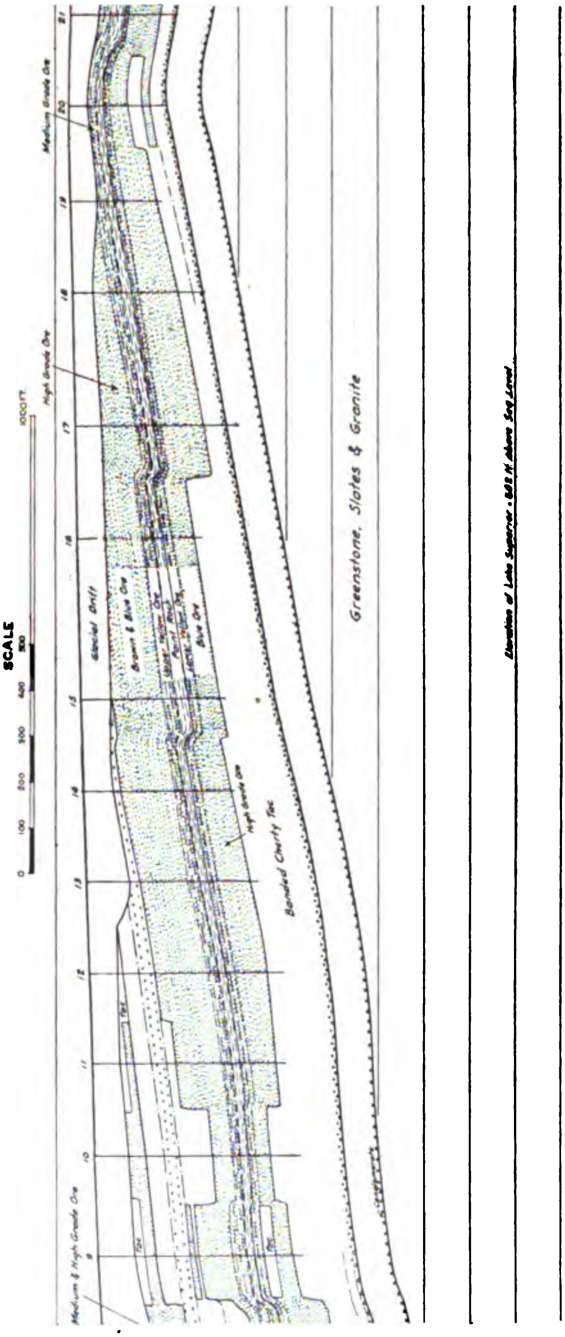


FIG. 6.—(*Continued.*)

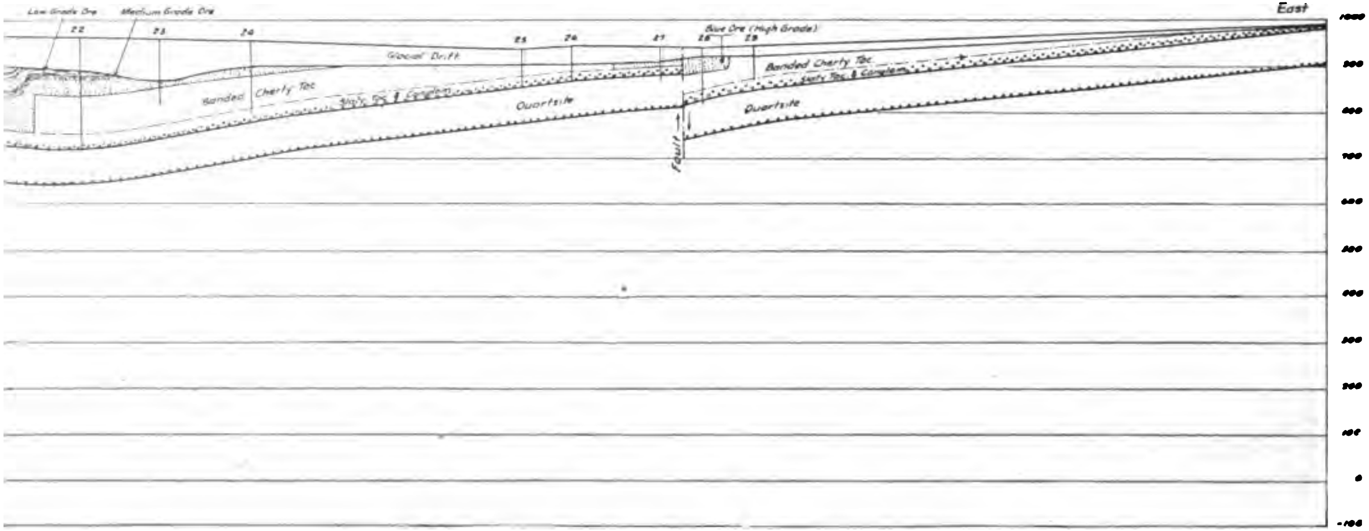


FIG. 6.—(*Continued.*) LONGITUDINAL CROSS-SECTION OF A LARGE TROUGH OREBODY SHOWING ORE DERIVED FROM ALL HORIZONS OF THE IRON FORMATION.

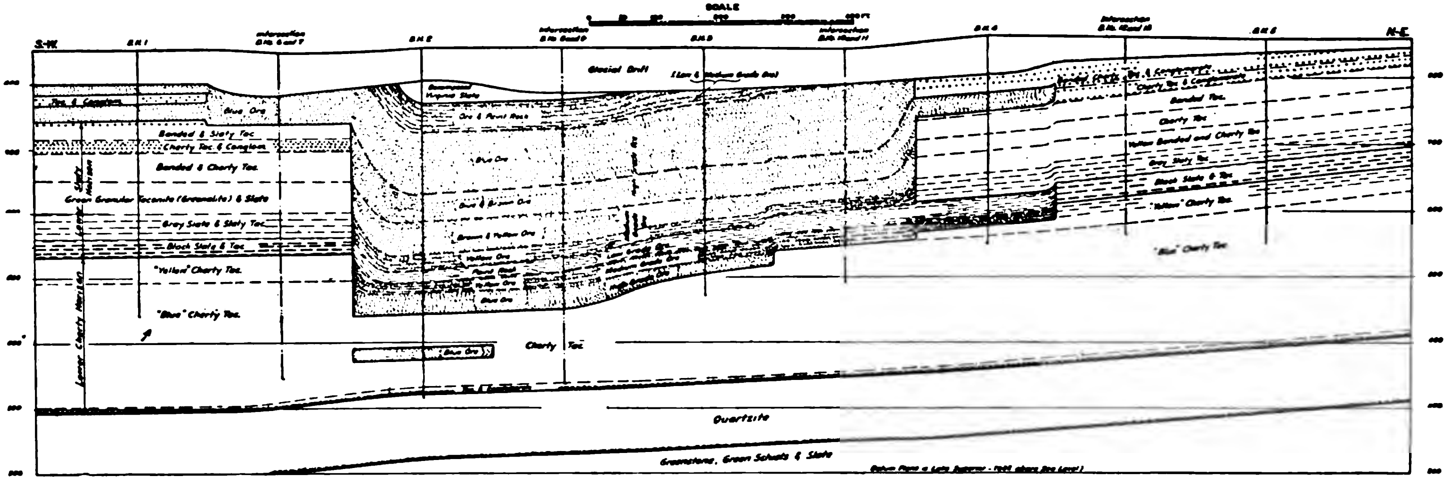


FIG. 7.—TRANSVERSE CROSS-SECTION OF A TROUGH OREBODY SHOWING ORE DERIVED FROM THE DIFFERENT HORIZONS AND TROUGH STRUCTURE OF OREBODY.

half of the Lower Slaty Horizon contains the most slate, the bottom 30 ft. or so being almost pure slate. But the upper half is slaty or banded in structure rather than in composition of the rock layers. The subdivisions are made by eye rather than on any chemical basis or microscopic examination.

The Upper Slaty Horizon is composed about half and half of slate layers and greenalite lenses. The upper part approaching the Virginia slate is the more slaty. It is separated from the Virginia slate by a layer 10 or more feet thick of a carbonate rock, probably an impure limestone.

Two cross-sections, Figs. 6 and 7, are presented to show in detail the four main subdivisions, the minor subdivisions of each, the relations of the orebodies to these horizons and the kind of ore derived from each.

Fig. 6 shows an east-west cross-section, looking north, through the Adams, Hull, Nelson, Leonidas orebodies just west of Eveleth. It is the best cross-section that has been or can be made from present explorations, so far as the author knows. The location of the section is shown on the accompanying map, Fig. 11, and is approximately at right angles to the strike. It is developed from drill-hole classifications, as can be seen, but practically the entire section of ore is being developed now by open-pit and underground workings. Because this cross-section is taken along the longitudinal axis of the ore trough, the trough structure does not show. In the area of drill holes 6 to 8 inclusive, a north-south trough is tributary to the east-west trough through which this cross-section is taken, and the trough structure is apparent. The gentle warping of the entire series can be noted. The principal warping is at right angles to the strike, however.

The relations between the different horizons and their derivative ores are so evident from the cross-section that a detailed description need not be given. In the area of drill holes 2 to 15 inclusive, the taconite in the Lower Slaty and Upper Cherty Horizons is so badly altered that classification is very difficult.

A few features may call for comment. The black slate indicated at the base of the Lower Slaty Horizon is the so-called "Intermediate" Slate, which makes the characteristic paint rock layer of the typical Mesabi orebody. The orebodies first developed on the Range were located in that part of the formation shown between drill holes 15 and 24; therefore, it is easily seen that the typical orebody had five layers—upper and lower blue ore layers, upper and lower yellow ore layers, and an intermediate paint rock layer.

Interbedded in the Upper Cherty Horizon and the top of the Lower Slaty Horizon are distinct conglomerates. At the base of the Upper Slaty Horizon there is also a fine interbedded conglomerate. The Upper Cherty conglomerate is continuous from the far eastern part of the Range to the western part. It was recognized in drill cores only a few years ago by W. H. Crago, head of the exploration division of the Mining Engineering Department of the Oliver Iron Mining Co. All of the earlier drill cores have not been reëxamined for it as yet, but it has been correlated extensively in different parts of the Range by F. B. Cronk and the writer. The cross-section in Fig. 6 shows a maximum development of interbedded conglomerate. It is not as thick either on the western or eastern end of the range. On the eastern part it is distinctly developed in both Upper Cherty and upper part of Lower Slaty Horizons, but in the west central part of the Range only the Upper Cherty conglomerate has been recognized, and it is quite thin. This section (Fig. 6) shows practically a maximum thickness of iron-formation. The Lower Slaty Horizon is abnormally thick (260 ft.) whereas the average thickness is only about 150 ft. In the central and western part of the Range, the Upper Cherty Horizon is exceptionally thick, the slaty layers of the upper part of the Lower Slaty Horizon being replaced by cherty and banded taconite. Evidently, more muds were deposited with the iron-formation in the east central part of the district than in the central and western parts of the district. Perhaps the underlying rocks outcropping to the north are accountable for this. From Mt. Iron east to Aurora, large areas of greenstones and slates lie to the north of the iron-formation. If the original shore line of the sea in which the iron-formation was laid down occupied approximately its present position, the weathering and erosion of these rocks contributed muds and argillaceous sediments to the sea water contemporaneous with the deposition of iron.

Fig. 7 shows a cross-section across a typical trough orebody in the Virginia district. The Upper Cherty and Slaty Horizons have been eroded from the sides of the rockwalls of the orebody, but all horizons from the Virginia slate down are represented in the orebody. The typical trough structure is well shown.

Records and Subdivisions of Drill Holes in Different Districts

Fig. 8 shows records and subdivisions of drill holes in the Nashwauk, Hibbing, Virginia, McKinley and Aurora districts, comprising the territory from the west central to the eastern part of the range. These records all show the same major divisions, though varying in the minor subdivisions and dimensions. The interbedded conglomerate in the Upper Cherty Horizon is persistent in all the districts.

Ores Derived from the Different Horizons

The characteristic ore derived from the Lower Cherty Horizon is a coarse "blue" high-grade ore. It contains practically no paint rock seams. About 30 or 40 ft. of the top of this horizon is a brown or yellow mottled cherty taconite (originally a greenalite rock) containing some slate seams in its upper measures, and this layer makes a yellow ocherous ore of medium grade. At the base of the Lower Cherty Horizon just

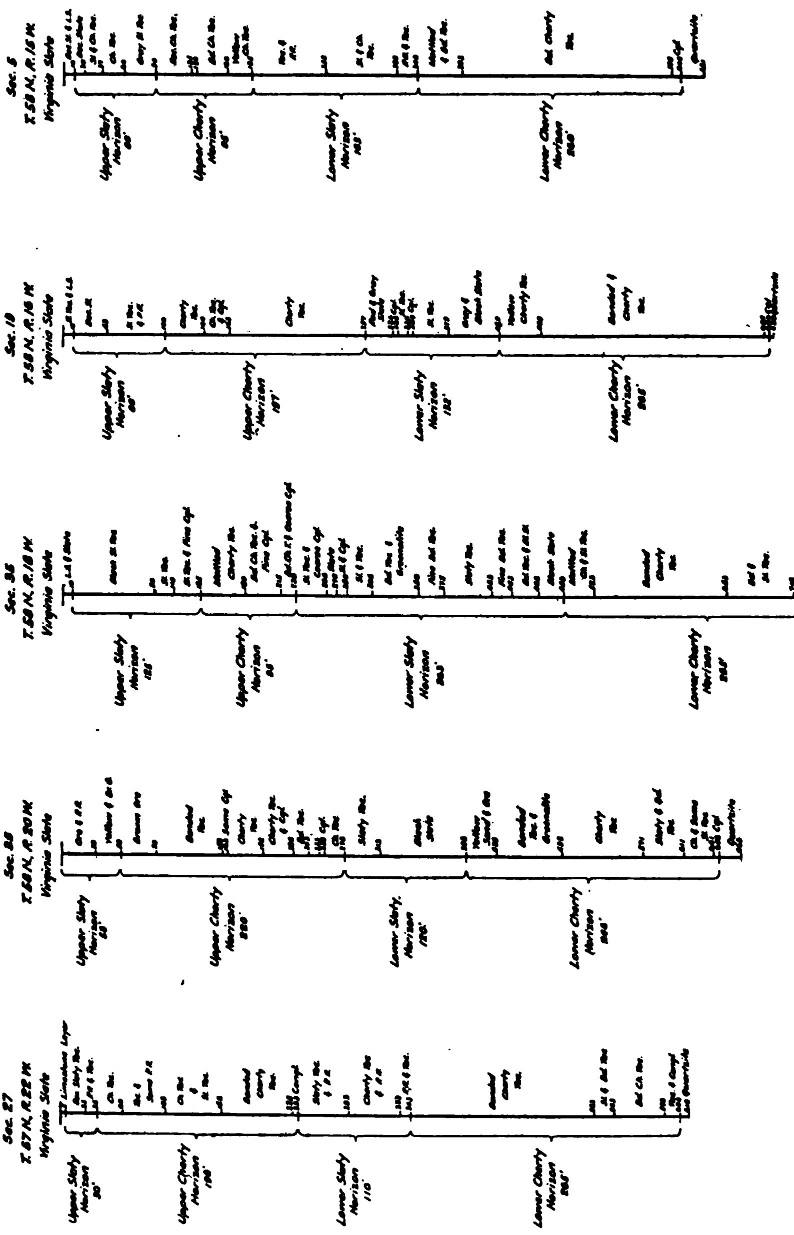


FIG. 8.—CHART SHOWING RECORDS AND SUBDIVISIONS OF DRILL HOLES IN DIFFERENT DISTRICTS ALONG THE MESABI RANGE.

above the basal conglomerate is a layer of fine slaty taconite (see Fig. 5) which makes a yellow ocherous ore. With the exception of these top and bottom layers, the Lower Cherty Horizon makes a "blue" high-grade ore. As used here, high-grade ore means ore averaging above 59 per cent. dry iron, medium-grade averaging 55 to 56 per cent. dry iron, and low-grade averaging about 50 per cent. dry iron.

The characteristic physical feature of ores from the Lower Slaty Horizon is their finely banded and slaty texture. As previously indicated, the black slate at the base makes the paint rock layer so prominent in every typical orebody. This material is not a commercial ore. It is highly aluminous and contains 20 per cent. or more of moisture. The gray slate and greenalite and slate above this black slate (Fig. 7) make a medium-grade yellow and brown ore, the yellow ore being quite aluminous. The banded-cherty and banded-slaty taconite of the top of the Lower Slaty Horizon make a very fine-grained blue and brown ore of high grade. It was this kind of ore which was so objectionable to furnace men because of excessive fines, in the early days of the Mesabi Range exploitation.

The Upper Cherty Horizon makes a high-grade coarse blue ore, in all of the large well-concentrated bodies. It is indistinguishable in texture from the blue ore of the Lower Cherty Horizon.

In some orebodies toward the south side of the formation, such as the Morton Mine in the Hibbing district, the Duncan in the Chisholm district, the Leonidas in the Eveleth district and the Schley and Hobart in the Gilbert district, the ore in this horizon is somewhat sandy and cherty, due in part to incomplete concentration and in part to secondary silica deposited by ground waters.

The Upper Slaty Horizon generally makes a low-grade non-merchantable ore. The one known exception to this is shown in Fig. 7, in which orebody most of this ore probably will be merchantable. This orebody, however, is one of the most highly concentrated on the Range. In most orebodies which have this Upper Slaty member, the material from it is a soft plastic paint rock with decomposed chert and greenalite layers. It resembles very much the so-called "Intermediate" paint rock layer.

From the cross-section, Fig. 6, it is evident that all orebodies will not contain all of these horizons or layers. The unit of land subdivision is a 40-acre tract and many mines occupy one or a part of one such tract. If a mine is located near the quartzite outcrop, most of the upper horizons will be eroded away, and as the mine location approaches the Virginia slate outcrop more of the upper layers will be found in the orebody.

RELATION OF OREBODIES TO FOLDING AND FRACTURING OF THE IRON-FORMATION

In the *Engineering and Mining Journal* articles above referred to, the author stated that the data then at hand indicated that the orebodies

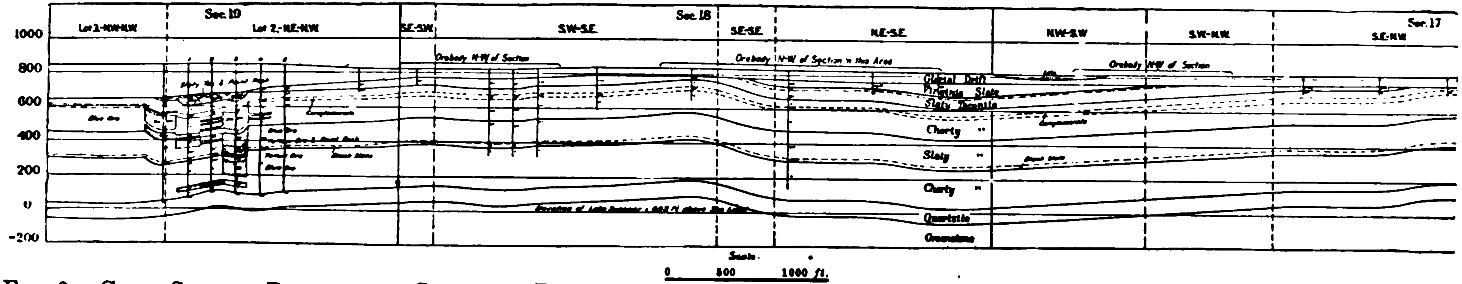


FIG. 9.—CROSS-SECTION PARALLEL TO STRIKE OF RANGE, SHOWING SUBDIVISIONS OF IRON FORMATION AND RELATION OF OREBODIES TO GENERAL STRUCTURE.

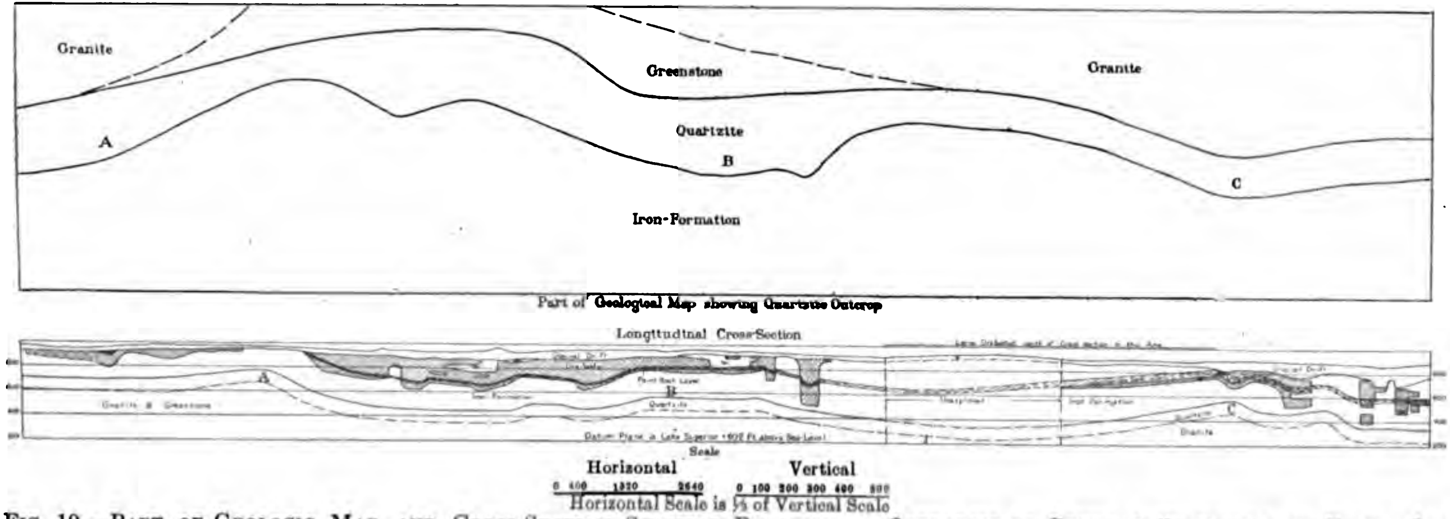


FIG. 10.—PART OF GEOLOGIC MAP AND CROSS-SECTION SHOWING RELATION OF OREBODIES TO GENERAL STRUCTURE OF ENTIRE IRON FORMATION IN HIBBING-CHISHOLM DISTRICT.

have formed in places where gentle folding and warping of the iron-formation had fractured it considerably, allowing easy access and circulation of ground-waters. Evidence along this line has been assembled since the publication of those articles and in every case where the exploration data is complete enough it has been found that the orebodies occur where the whole formation has been warped. In the eastern part of the district (Virginia and eastward) the orebodies are on the crests of gentle anticlines. Fig. 9 shows a cross-section parallel to the strike of the formation in T. 58 N., R. 16 W., location of which is shown on the map in Fig. 11. This section is taken well to the south side of the formation and only one orebody reaches it. But the locations of other orebodies northwest of it are shown on the cross-section, and in each case where the exploration data is complete enough to show it, the orebody is located on the axis of an anticlinal flexure or combined anticline and syncline.

In the central part of the Range, great broad flexures rather than merely localized ones seem to have determined the locations of the orebodies. The formation was very generally cracked up and the broad structural basins directed the flow of underground waters. Fig. 10 shows a structural cross-section taken midway between Virginia slate and quartzite outcrops approximately parallel to the strike, through the Hibbing-Chisholm districts. A part of the quartzite outcrop to the northwest is also shown. Three prominent anticlines, *A*, *B* and *C*, with two intervening synclines, are shown. The cross-section is taken about a mile south of the quartzite outcrop. It has been published previously in the *Engineering and Mining Journal*, Aug. 7, 1915. It shows that the orebodies occur quite continuously over both crests of anticlines and troughs of synclines, indicating a very general fracturing of the formation, vigorous circulation of ground-water and consequent complete alteration and concentration of iron-formation. These major flexures can be determined only by such correlation and drawings as are shown in the two cross-sections, Figs. 9 and 10, but the minor flexures within the larger ones often can be observed in the field.

SPECIAL FEATURES

Special features of the Mesabi Range may be of interest and deserving of inclusion in this paper.

FAULTS

The two principal faults known to date on the Mesabi Range are those known as the Biwabik and Alpena faults, both of which were described in the *Engineering and Mining Journal*, July 24, 1915. Mention is made of them here only because they have been followed further since that time. Fig. 11 shows the location of both. The Biwabik fault has been traced to the NW of SE, Sec. 5, 58–15, where it practically disappears. It is a hinge-type gravity fault, the south side of which has been depressed. The greatest throw is at its west end at the Biwabik Mine, Lot 4, Sec. 2 and Lot 1, Sec. 3, Tp. 58 N., R. 16 W., where the vertical displacement exceeds 200 ft. The underlying greenstone is faulted up against the ore, though the faulting probably occurred prior to the formation of the ore. Fig. 12 is a cross-section of the Alpena orebody north of Virginia, showing the largest fault known on the Range. The location of the cross-section is shown on Fig. 11. As indicated, the strike of the fault is approximately north and south. It is a faulted thrust-fold, the probable development of which is shown by Fig. 13.

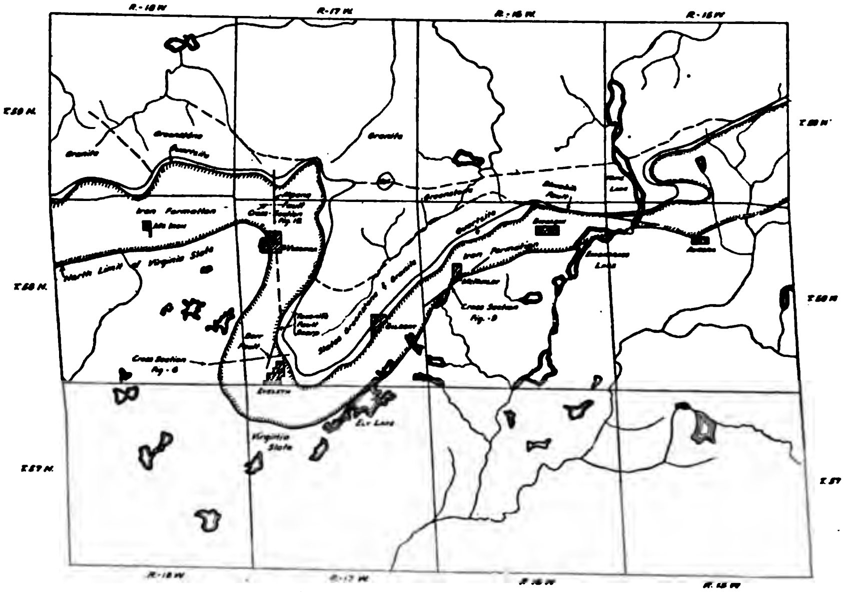


FIG. 11.—PLAT SHOWING AREA OF EAST CENTRAL MESABI RANGE.

The genesis of the fault is directly connected with the gentle uplift or crustal warping which produced the large Z-shaped bend in the Range known locally as the Virginia "horn." This was discussed also in the *Engineering and Mining Journal,* July 24, 1915. It is repeated here because since that series, a fault (shown on the east end of cross-section in Fig. 6) has been discovered by correlation of drill holes. The location of this fault, which may be called the Dorr fault from the property on which it is located, is shown on the map, Fig. 11, and is of the same type (reverse or thrust) as the Alpena fault. It is almost directly south of the Alpena. Between the two and about ¾ mile north of the Dorr fault is a taconite bluff, the east side of which is a very steep wall, undoubtedly a fault scarp. It is of the same type as the Dorr and Alpena faults.

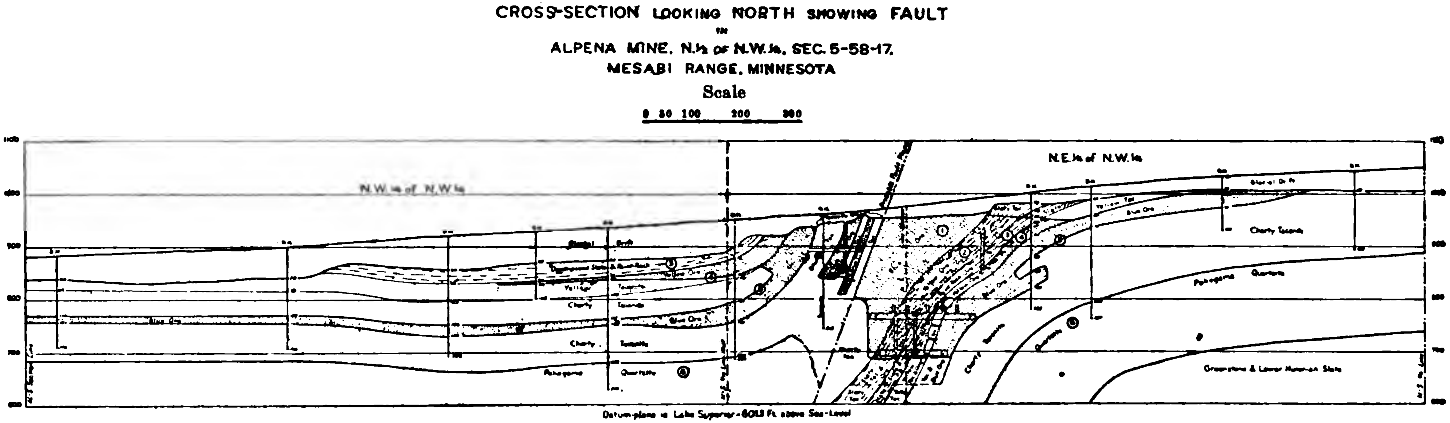

CROSS-SECTION LOOKING NORTH SHOWING FAULT
IN
ALPENA MINE, N.½ OF N.W.¼, SEC. 5-58-17.
MESABI RANGE, MINNESOTA
Scale
0 50 100 200 300
1100
1000
900
800
700
600
N.W.¼ of N.W.¼
N.E.¼ of N.W.¼
Blue Ore
Yellow Taconite
Cherty Taconite
Pokegama Quartzite
Greenstone & Lower Huronian Slate
Datum-plane is Lake Superior-601.3 Ft. above Sea-Level

FIG. 12.—SHOWING ALPENA THRUST-FAULT.

Although we have as yet no complete exploration data, it appears that the Alpena and Dorr faults and the intervening escarpment are one continuous fault produced by the crustal movements which caused the Virginia "horn." This probable connection is indicated on Fig. 11. However, it is possible that the Alpena fault is entirely separate from the Dorr fault.

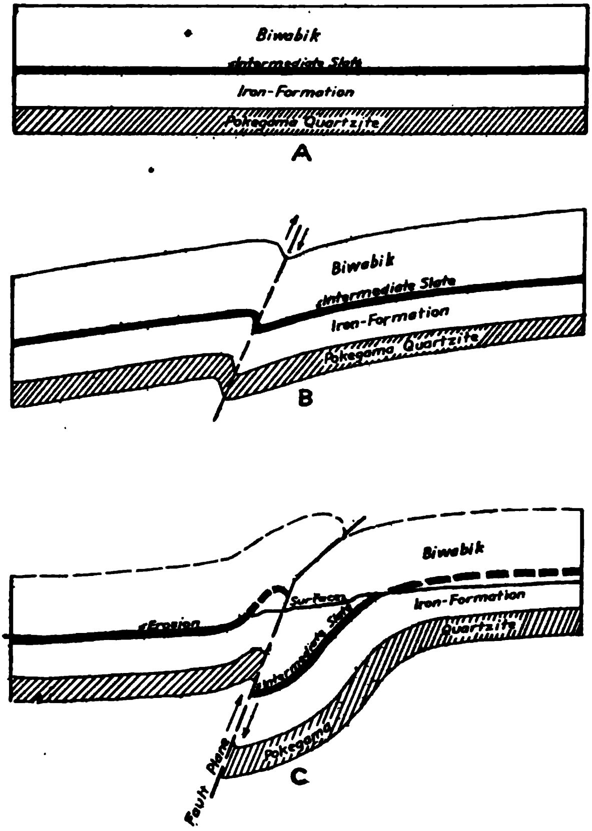


FIG. 13.—SHOWING PROBABLE DEVELOPMENT OF THE ALPENA THRUST-FAULT.

Thickness and Dip of the Iron Formation

The average thickness and dip of the whole iron-formation in different parts of the Range may be of interest. So far as they have been determined from the present very extensive explorations, they are shown in the accompanying table, tabulated by Ranges.

It will be noted that west of the Alpena fault the dips are quite low and uniform, while east of it they are higher and rather irregular. From this evidence and that of the Alpena-Dorr fault, it appears that the disturbance caused probably by the intrusion of the great mass of Duluth

	Average Dip, Degrees	Thickness, Feet
T. 56 N.—R. 24 W.	6	520
T. 56 N.—R. 23 W.	3 to 8	475 (approx.)
T. 57 N.—R. 22 W.	5	615
T. 58 N.—R. 21 W. T. 57 N.—R. 21 W.	5	590
T. 58 N.—R. 20 W. T. 57 N.—R. 20 W.	4½	650
T. 58 N.—R. 19 W.	3 to 7	660
T. 58 N.—R. 18 W.	3	630 west of Alpena fault
T. 58 N.—R. 18 W. T. 58 N.—R. 17 W.	6 to 9	755 east of Alpena fault
T. 57 N.—R. 17 W.	10 to 15	
T. 58 N.—R. 16 W.	12	650
T. 58 N.—R. 15 W.	25	530 (Secs. 5 and 6)

gabbro between Lake Superior and the eastern Mesabi range tilted the eastern part of the Range considerably; that the Alpena fault took up most of the crustal shortening in the Upper Huronian series due to the compression from the east; and that, because of the faulting, the sedimentary series west of it was relatively undisturbed.

The thickness is seen to be greatest in the vicinity of Eveleth and thinnest on the extreme eastern and western ends. The average of the figures given is about 620 ft.

Volumetric Shrinkage in Solid Taconite

Recent structural study has shown that the alteration which the original iron formation has suffered has produced a volumetric shrinkage even where the iron formation still retains its solid condition. However, in such solid taconite, hard layers of high-grade iron ore are interbanded with chert layers, the whole mass being firmly cemented together. Drill holes in fresh unaltered iron-formation alongside of holes in altered but still solid taconite show this fact to exist. Such a case is shown in comparing the depths of formation below the Lower Slaty Horizon in holes 1 and 2 with those in holes 4 and 5 in the orebody in Sec. 19, Fig. 9; also holes 1 and 2, Fig. 6.

This fact is a very important one to the mining engineer in the district in working out the structure of orebodies. The knowledge of it will aid him to establish more accurately the correct structure of certain ore layers. It explains the apparent lack of parallelism between different members of the iron-formation, which cross-sections often indicate unless this fact is known and applied.

Post-Algonkian Conglomerates on the Upper Huronian Series

In the *Engineering and Mining Journal,* July 17, 1915, the writer called attention to at least one conglomerate, and possibly two, on the top of the

Biwabik iron-formation, older than the so-called Cretaceous conglomerate, beds and remnants of beds of which are found on most of the orebodies west of Eveleth. Since that time he has observed in two mines a conglomerate capping the orebody, containing large boulders which themselves were composed of the typical Cretaceous conglomerate. These conglomerates were overlain by layers of plastic shale and muds. Undoubtedly these are local lakebed or stream bed conglomerates and muds, intermediate in age between Cretaceous and Pleistocene.

Newly Discovered Fossil Remains in the Cretaceous (?) Shale

The conglomerate and shale found on top of many orebodies were correlated by the U. S. Geological Survey as Cretaceous from the fossil remains of bivalves and some teeth and vertebræ of the Mosasaur found in the shale. During the past year, in the Canisteo pit at Coleraine, one almost perfect fossil and part of others of Ammonites were found. A reproduction is shown in Fig. 14. As nearly as the writer and associates can correlate these specimens, they belong in the Jurassic and not in the Cretaceous. They show no such complicated sutures nor such ornamentation as is characteristic of the Cretaceous Ammonites. The Specimen *A*, Fig. 14, is about 15 in. in diameter and 3 in. thick. On Specimen *B*, which fits into the cast *C*, and *C* also, can be seen minute veins, but no evidence of complex sutures. It is possible, of course, that this Jurassic form lived over into the Cretaceous, but the numerous Saur remains the writer has seen could hardly have belonged to a reptile as large as the Mosasaur. They seem more fitted to a smaller creature, perhaps not over 10 ft. long, rather than to one 30 or 40 ft. long. May they not belong to a Jurassic saur, one less fully developed than the Cretaceous Mosasaur? Is it not possible, or even probable, that the conglomerates and shales on the Mesabi Range which have been called Cretaceous really are of Jurassic age and formerly were connected with the extensively developed Jurassic shales of Northwestern Minnesota? Northern Minnesota is such a well-developed peneplain that it is difficult to imagine a Jurassic ocean covering any part of it without reaching at least the foothills of the Giants or Mesabi Range.

Virginia Slate, Iron-Formation Contact

Because of possible value in connection with a revision of the correlation of the Huronian series in the Lake Superior district, the writer wishes to append to this paper the following record of observations of the relation of the Virginia slate to the underlying iron-formation, particularly because it is not in accord with statements as to this relation made in *Monograph* 52 of the *U. S Geological Survey.* The latter publication states that at the top of the iron-formation and the base of the Virginia

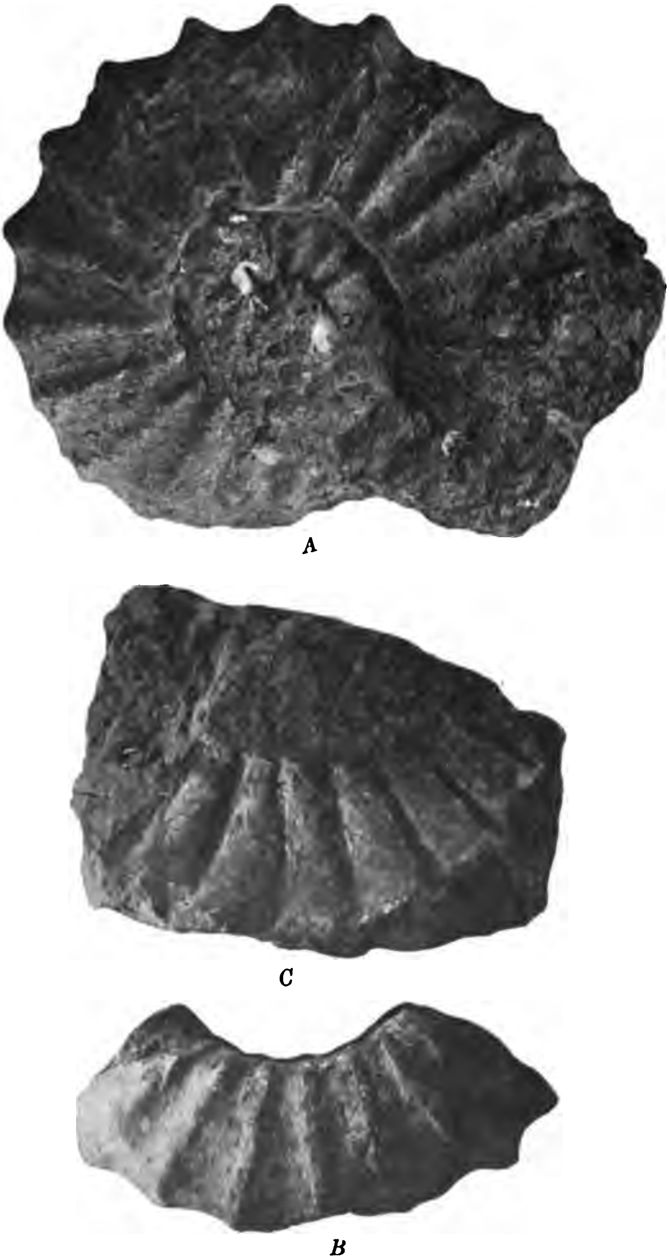


FIG. 14.—FOSSILS FOUND AT COLERAINE.

slate, there is a horizon perhaps a few hundred feet thick which is one of gradation, in which layers of iron-formation and slate alternate, and that "the layers of slate are found well down in the iron-bearing formation, and layers of the iron-bearing formation are found well up in the slate" (page 174, *Monograph* 52, *U. S. Geological Survey*).

An examination of cores from a great number of holes which penetrated through the Virginia slate into the underlying iron-formation has failed to substantiate this statement. Cores from two drill holes which penetrated between 500 and 600 ft. of Virginia slate and the entire underlying iron-formation to quartzite, and from scores of other holes, failed to show the presence of a single layer of greenalite in the true Virginia slate. Several thin layers of a carbonate rock (probably calcium carbonate) and a few crystals of iron carbonate were observed. In the iron-formation proper a very few bands of a carbonate rock were discovered. As shown on Fig. 5 of this paper, the upper horizon of the iron-formation is a slaty horizon in which layers of dark slate and greenalite alternate. At the top of this horizon and separating it from the true Virginia slate (which is a dense dark gray or black slate), is a layer several feet thick of calcium carbonate, amorphous or very finely crystallized. This layer is mentioned on page 171, *Monograph* 52, *U. S. Geological Survey*. Wherever drill holes have penetrated through the Virginia slate all along the Range, this carbonate has been found immediately beneath it. In a few holes, cores of which were examined very carefully, a small amount of conglomeratic material was found in the upper part of this carbonate layer.

Although the average thickness of the iron-formation in adjacent areas is quite uniform, as shown in the table of average dips and thicknesses, there are marked irregularities within short distances. Differences in total thickness of iron-formation of 20 to 50 ft. in drill holes ¼ mile apart are common. Two holes, 1¼ miles apart show a difference in total thickness of 121 ft.; two holes 2½ miles apart show a difference of 184 ft. Not enough close subdivision and correlation work has yet been done to determine whether such differences are due to initial deposition or erosion from the top of the iron-formation. There is so much interbedded fine conglomerate in the upper horizons of the iron-formation that we know definitely that this part of the formation at least was deposited in very shallow water. It is not at all improbable, therefore, that prior to the deposition of the Virginia slate, the iron-formation may have been raised above water, and its upper surface somewhat eroded. The variation in thickness of the Upper Slaty Horizon gives support to this idea. No marked unconformity between the two formations can be established, however. They are conformable stratigraphically, as far as now known, but the significant fact of the absolute lack of any known greenalite (so far as revealed by many years of observation of thousands of feet of drill cores by the director of explorations of the Oliver Iron Mining Co.) in the Virginia slate and its marked prevalence immediately beneath the Virginia slate seems to call for such a pronounced change in conditions of deposition as to demand some time interval between the two. The universal prevalence of the calcium carbonate layer with some conglomerate between the two gives further support to this idea of a time interval.

These facts, minor though they may be, are presented here in the hope that they may be of some value to the geologists who are engaged in the revision of the correlation of the Huronian Series of the Lake Superior District.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

BY MAX ROESLER, PH. D.,* SANTIAGO DE CUBA, CUBA

(New York Meeting, February, 1917)

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[*] Geologist, Spanish American Iron Co. and Juragua Iron Co.

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I. INTRODUCTION

The following article concerns the geological occurrence of the iron-ore deposits on the south coast of Cuba. The article is based on a detailed field study, made in the hope that some information would be gained which might be of value in the search for further orebodies, or in the economic development of the ore already found.

Location

The iron-ore deposits of the Firmeza district lie near the coast, in the southeastern part of Cuba, in the Province of Oriente. The orebodies of this district form part of a belt of deposits that extends from Sigua, 25 miles east of Santiago, to Sevilla, 5 miles east of Santiago, and lies on the seaward slope of the Sierra Maestra range of mountains.

This range roughly parallels the coast in the southern part of Oriente Province. Its crest is about 6 miles north of the mines near Firmeza. The town of Firmeza lies 9 miles east of Santiago, and about 2½ miles from the Caribbean Sea. The mines included in the Firmeza district are the Ocania Mine and a group of mines that extends from West Five Mine to the Concordia Mine. The elevation of the mines is from 400 to 1,000 ft. above sea level. Their exact location is shown on the map (Fig. 1) which is a copy of a map furnished by the Juragua Iron Co.

Immediately east of the Firmeza district lies the Daiquiri district of the Spanish-American Iron Co.

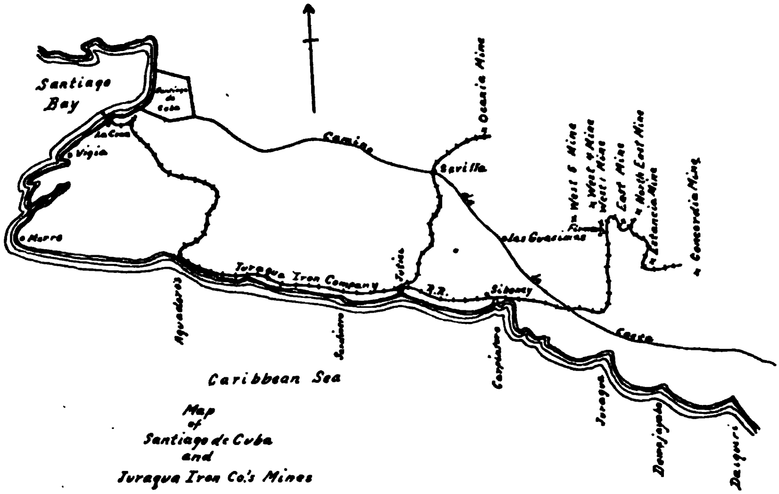

FIG. 1.

SCOPE OF WORK AND ACKNOWLEDGMENTS

Detailed examinations have been made by J. P. Kimball,[1] A. C. Spencer,[2] and J. F. Kemp,[3] and many others have contributed to our knowledge of the deposits. The writer has made use of all the available information and will endeavor to give full credit for it.

The present article has been written under the first award of the S. F. Emmons Memorial Fellowship, and the writer takes this occasion to thank the committee in charge of the fellowship for their assistance and criticism. Thanks are also due to the Juragua Iron Co. for permission to make the examination, for the use of their maps, and for their kindness in giving access to all the available data and facilities. The Spanish-American Iron Co. also made possible the visiting of neighboring

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[1, 2, 3] See Bibliography at end of paper.

properties and most grateful acknowledgment is hereby made to the officials and staff of both companies.

The Geological Faculty of Yale University, and Professor C. P. Berkey of Columbia University, have been freely consulted on different phases of the problem, and their advice has been of much service.

History and Mining

The Firmeza orebodies have been worked continuously since 1884, except for a short period during the Spanish-American war, and the total production of the mines, from 1884 to 1913 inclusive, has been 6,776,171 tons.[4] The mines of the Firmeza district belong to the Juragua Iron Co., which is now controlled by the Bethlehem Steel Co., though it was at one time also allied with the Pennsylvania Steel Co.

The mining of the ore and the stripping of the overburden is done by steam shovel. Where the orebody is small or intimately associated with waste, it is found advantageous to lease the workings to contractors and pay for the ore on the basis of tonnage and grade, delivered in mine cars.

The ore from the mines is put through the crusher at Firmeza, from which the major part is shipped direct to La Cruz, the company's shipping point on Santiago harbor. A small part of the output, high in sulphur, is roasted in the valley south of Firmeza, near Siboney, before it is shipped to La Cruz.

II. TOPOGRAPHY AND ITS INTERPRETATION

The topography of the Firmeza district falls naturally into three main divisions which have a lateral extent parallel to the sea coast and the main range of the Sierra Maestra,—that is, in an east-west direction. Beginning at the coast there is a range of terraced cliffs rising to uniform height of about 300 ft. The terraces all face the south, to the sea, and appear to be of wave-cut origin, excavated in flat-lying coral limestone. There is no beach, the sea beating directly against the limestone cliffs, except at the mouth of the Rio Carpintero at Siboney, where there is a small beach made up of coral fragments and an arkosic sand in which feldspar predominates.

Back of the terraces there is a fairly gentle landward slope to a silt-covered, flat, east-west valley. In this valley the streams from the mountains are generally lost in lagoons and swamps. Some of the larger ones, such as the Rio Carpintero, find their way to the Caribbean through comparatively narrow gorges cut in the cliffs that border the sea. These streams appear to be of an intermittently torrential type. In the dry season they show very small streams of water flowing through stream beds covered with boulders up to several tons in weight.

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[4] D. B. Whitaker: *Engineering and Mining Journal*, vol. 97, p. 677 (1914).

North of this valley rise the foothills of the Sierra Maestra—sharp, steep hills, covered as a rule with dense forest growth. They are connected with the main range of the mountains by ridges that have steep slopes and narrow crests. The connecting ridges are less heavily forested than the foothills but are covered with high grasses and an occasional grove of trees. The mountains themselves are thickly covered with pines and rise to an elevation of about 3,500 ft.

The limestone cliffs, with their sea-cut terraces, have been interpreted as evidence of periodic movements of the island, but it seems to the writer that the widespread occurrence of a terrace about both Jamaica and Cuba at the same elevation above sea level indicates a movement of the sea surface, rather than of the land.

In speaking of the Seboruco (the local name for the coastal limestone) R. T. Hill says:[5]

> "Nowhere have I seen the elevated reef rock folded or otherwise disturbed except by the gently sloping coastward inclined elevation it has undergone.——The Seboruco as a whole represents a recent and uniform elevation of the whole periphery of the island——."

The following terrace elevations have been taken from the report of C. W. Hayes, T. W. Vaughan, and A. C. Spencer,[6] and arranged in tabular form by the writer:

Havana, Feet	Matanzas, Feet	Gibara, Feet	Baracoa, Feet	Manzanillo, Feet	Santiago, Feet
4–5	5–6	5–20	5–6	5–20	20
10–15	30	40	90	100	100
100	140	100	250	200	280
200	200	150–180	....	....	....
......	300	....	....	....	....

R. T. Hill[7] in the report on Jamaica has given less definite figures for the terrace elevations, and the following conclusions:

> "In general the old reef rock of Jamaica consists of three distinct formations, occurring at three levels, 70, 25, and 10 ft. (or less) respectively. From the persistency of these three levels on the north, east and southwest end of the island, it is evident that their present position above the water is due to continuous epeirogenic elevation after the present outlines of the island had been chiefly defined."

For Hayti and Porto Rico no exact data are available to the writer.

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[5] R. T. Hill: Notes on the Tertiary and Later History of the Island of Cuba. *American Journal of Science*, Ser. 3, vol. 48, p. 203 (1894).

[6] *Op. cit.* (in Bibliography), pp. 18, 19.

[7] R. T. Hill: Geology of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, pp. 92–100 (1899).

The data quoted show that on the north, northeast, and southeast coast of Cuba, and on the north, east, and southwest coast of Jamaica, there is a marine terrace from 5 to 20 ft. in elevation. It seems to the writer that this evidence might suggest that, at least in part, the differential movements of land and sea in Cuba have been due to movement of sea level rather than of land surface. The slight variations in elevation would then be explained by unequal erosion of the gently sloping surface of marine planation. The slight seaward slope of the coast limestone, in the neighborhood of Santiago, is no greater than is to be expected of a near shore surface of marine planation.

There is further evidence of emergence of the island, in respect to sea level, in the tuff-limestone found at an elevation of about 1,400 ft., and the steep-sided, sharp, mountain topography points to fairly recent rejuvenation.

The silt-covered, east-west valley, in which the streams are aggrading, suggests that the last movement of land with respect to sea level has been a downward one, and this is in accord with the evidence of the harbors of Santiago and Guantanamo, which have been interpreted as bays formed by drowning.[8]

The topography seems to be almost entirely independent of the lithology. An exception to this is the coarse crystalline marble that forms a capping to many of the foothills. Apparently the massive marble offers greater resistance to tropical weathering than do the fine-grained, diabasic or dioritic rocks which marmorized it, for the arroyos are cut into the latter. The occurrence of marble as a capping is too frequent to be entirely fortuitous.

III. PETROLOGY

SEDIMENTARY ROCKS

Description

The sedimentary rocks are represented in the Firmeza district entirely by limestones. At the coast, and rising in three sea-cut terraces to an altitude of about 350 ft., are recent limestones. According to the report of Hayes, Vaughan, and Spencer,[9] this rock is

> "replete with the remains of numerous species of corals which are all, so far as examined, at present living in the surrounding Antillean seas."

There seems to be one rather striking point of difference between the higher landward terraces and the lower terrace bordering the Caribbean. This is, that while all three terraces contain coral remains, the landward terraces are almost entirely massive, while the seaward terrace is made up of more loosely cemented coral remains. On the land, where the limestone comes into contact with the underlying igneous rock, there are boulders of rock and of iron ore cemented by the lime, showing that erosion of the orebodies had begun before these coral rocks were formed.

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[8] Hayes, Vaughan, and Spencer, *Op. cit.*, p. 17.

[9] *Op. cit.* pp. 23–24.

In the foothills of the Sierra Maestra, which vary in altitude from 600 to 1,300 ft., there occur, usually as a capping on the hilltops, bodies of massive limestone now largely marmorized. Bedding is almost entirely lost, but where thinner masses have been involved in the volcanic rocks it is possible to discern a pitch toward the southeast at an angle of about 30°. The freshest pieces of this limestone, taken from boulders on the north side of the hill north of West Five Mine, show it to be a blue, dense, fine-grained limestone. The microscope shows no evidence of organic remains in the thin section, but only a granular aggregate of calcite.

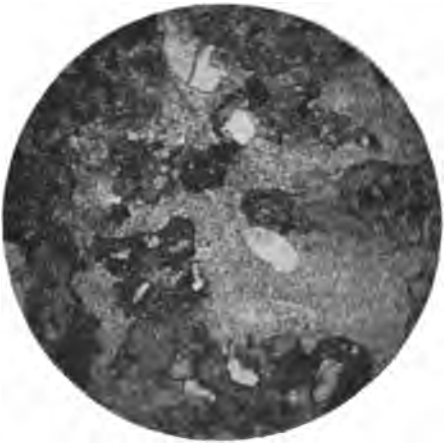

FIG. 2.—TUFF-LIMESTONE FROM RIDGE CONNECTING FOOT HILLS WITH MAIN RANGE OF THE SIERRA MAESTRA. Limestone—groundmass. Plagioclase fragments—white. Volcanic rock fragments—dark. × 18.

On one of the ridges connecting the foothills with the main range, at an elevation of about 1,400 ft., an unaltered limestone of unusual type was found. This rock has the deeply pitted surface typical of the weathered outcrops of all the limestone in this region. The color of the rock on a fresh fracture surface is blue-gray. Closer examination shows minute areas with a vitreous luster embedded in a granular calcite matrix, giving to the rock the appearance of a porphyry. A thin section of the rock shows that the crystals are fragments of plagioclase, of about andesine-labradorite composition. There are also angular fragments of a diabasic rock (Fig. 2). The fragments are all about 0.5 mm. in diameter, and their presence in the limestone shows that the sediment was formed at a time of volcanic activity on the adjacent land. The perfect freshness of the feldspars is probably due to the fact that the sea water was already saturated with calcium carbonate, and so was not chemically active in so far as the calcic feldspars were concerned. The ferromagnesian minerals of the diabasic fragments are entirely altered. This tuff-limestone seems to be very similar to that described by C. P. Berkey[10] among the "probably" pre-Tertiary rocks of Porto Rico.

Age

The nature of this rock shows that at least part of the volcanic activity was contemporaneous with the sedimentation. As the andesitic rocks accompanying this tuff-limestone are believed to represent the initial stages of the igneous cycle, the determination of the age of the sediment must also fix the age of the igneous rocks.

The age of these limestones and the associated volcanics has been a rather vexed question. J. F. Kemp speaks of an article by H. Wedding,[11] in which he places them in the Jurassic, in the horizon of Quenstedt's Beta, of the upper White Jura. This determination was made upon material furnished by G. W. Goetz. The only other definite information on this subject of age is that given by J. T. Singewald, Jr., and B. L. Miller.[12] They found a fossiliferous limestone in the Daiquiri district, and submitted some of the fossils to T. W. Vaughan, who correlated one of the species with the Cretaceous of Jamaica and determined the age of the limestone as "Mesozoic, probably Cretaceous."

W. Lindgren[13] says:

> "The idea of the geologists who have done the most work in this section seems to be that the lavas and tuffs and associated limestones are of Eocene age——."

R. T. Hill,[14] speaking of the volcanic clastics of the Jamaican Blue Mountain Series of Cretaceous age, says:

> "In Cuba these clastic rocks constitute the high divides of the Oriente——."

Experience during historic time shows that it would not be justifiable to infer volcanic activity in one of the islands of the Antilles on the basis of such activity in another. In the present case there is, however, the evidence of Cretaceous sedimentation in Jamaica determined by R. T. Hill, the direct correlation of the Daiquiri specimen with the Jamaican fauna by T. W. Vaughan, and R. T. Hill's reference to the clastic rocks of Oriente. E. T. Hodge has informed the writer that he found Comanche fossils among the pre-Tertiary.

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[10] C. P. Berkey: Geological Reconnaissance of Porto Rico. *Annals of the New York Academy of Sciences*, vol. 26, p. 20 (1915).

[11] H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. 12, No. 12, pp. 545–550 (June 15, 1892).

[12] J. T. Singewald, Jr., and B. L. Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, pp. 67–74 (1916).

[13] W. Lindgren and Clyde P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, p. 41 (1916).

[14] R. T. Hill: The Geology and Physical Geography of Jamaica. *Bulletin* of the Museum of Comparative Zoology at Harvard, vol. 34, p. 170 (1899).

IGNEOUS ROCKS

The igneous rocks of the Firmeza district form a natural series of differentiation products from a basic magma. They range from a fine-grained diabase to a highly quartzose aplite. J. F. Kemp[15] has given a discussion of the nomenclature used in former articles on the district.

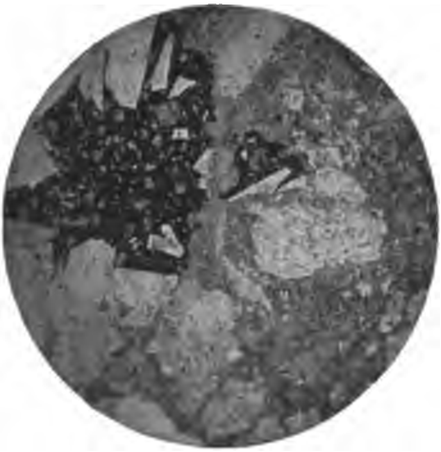

FIG. 3.—DIABASE PORPHYRY WITH INCLUDED FRAGMENT OF EARLIER VOLCANIC ROCK. WEST FOUR MINE. × 18.

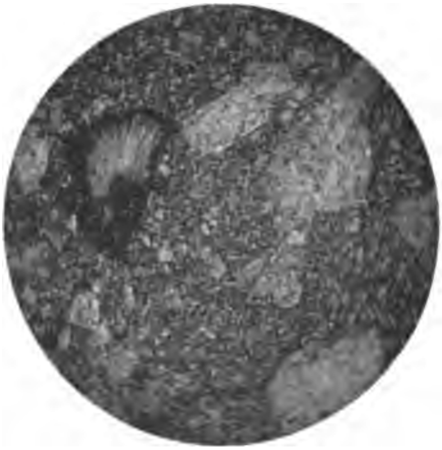

FIG. 4.—DIABASE PORPHYRY WITH EPIDOTE-FILLED AMYGDULE. FROM RAILROAD CUT OPPOSITE WEST ONE MINE. × 18.

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[15] The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, vol. 53, pp. 3–38 (1916).

For purposes of mapping, the writer divides the igneous series into four groups:

1. The diabasic rocks.
2. The dioritic group.
3. The granitic group.
4. The later dike rocks.

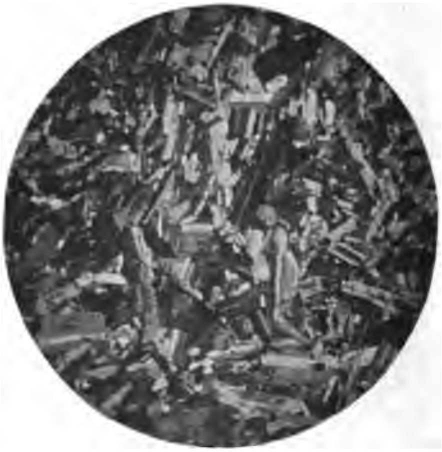

FIG. 5.—DIORITE FROM LEVEL 1 OF WEST FIVE MINE. CROS ED NICOLS. × 18.

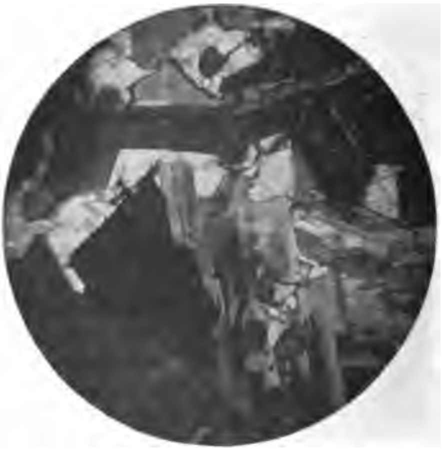

FIG. 6.—QUARTZ DIORITE FROM EAST MINE. THIS IS THE BASIC MEMBER OF THE GRANITIC SERIES. CROSSED NICOLS. × 18.

Diabasic Rocks

The diabasic rocks are fine-grained, as a rule, and porphyritic. They are the representatives of the original magma. Megascopically they are dense, dark rocks. Under the microscope they show as felty aggregates of plagioclase laths with interstitial ferromagnesian minerals. They are usually fragmental (Fig. 3) and frequently show epidote-filled amygdules (Fig. 4).

These diabasic rocks show a bedded structure in places, with intercalated limestones and are undoubtedly in part extrusives.

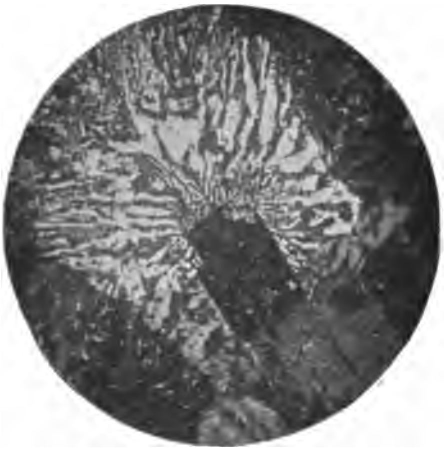

Fig. 7.—Granite Showing Micrographic Quartz Feldspar Intergrowth about Plagioclase Nucleus. From the Juragua Valley. Crossed Nicols. ×18.

Dioritic Rocks

The wall rock of many of the orebodies is a gray, fine-grained, and even-textured rock. Plagioclase, varying from labradorite to bytownite, is the most abundant mineral, with much hornblende between the feldspars. On the basis of the feldspars it is a gabbro, but in the older

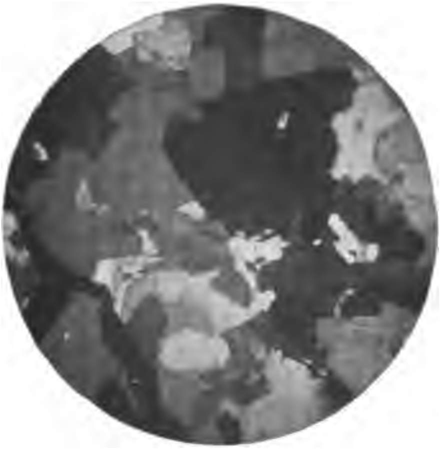

Fig. 8.—Specimen B from Dike in West Side of West Five Mine. Elevation 520 Ft. Note Corrosion of Plagioclase and Dominance of Quartz. Crossed Nicols. × 18.

classification the hornblende makes of the rock a diorite. Since that terminology has been adopted by earlier writers on the district it is accepted here. Fig. 5 shows a typical diorite.

The diorites and diabases are frequently found merging into each other without any visible contact. Both are the hosts of the orebodies.

Granitic Rocks

J. F. Kemp[16] has used the term granite for the quartz-bearing diorites and the true granites "to avoid all confusion of this rock with the diorites which are associated with the ore." That usage is adopted in this paper, and the scope of the term granitic rocks is enlarged to include the aplites and quartz porphyries. This inclusive usage seems justified by the fact that the quartz-bearing rocks form a group later than the ore-bearing rocks, though pre-mineralization.

All the rocks of this group are hornblende-bearing except the most acid aplites, and even the most acid show very little potash feldspar. The basic members of the series are gray, coarse-grained, feldspathic rocks (Fig. 6) while the acid members are made up of striking feldspar quartz intergrowths (Fig. 7), or else are fine-grained quartz aplites (Fig. 8). The extremely rapid variation in chemical composition is shown by three microscopic analyses of rocks from the same dike at different elevations and ascending order.

	Feldspar	Quartz	Ferromag.	Magt.
A..........	61.8	28.0	5.5	4.7
B..........	54.0	41.6	3.4	1.0
C..........	52.7	45.2	1.1	1.1

Later Dike Rocks

The entire mineralized area is cut by basic dikes that vary from basalt to andesite. They are post ore, but usually carry pyrite. A detailed description would serve no definite purpose in this connection.

IV. AREAL GEOLOGY

ROCK TYPES FOUND IN THE DISTRICT

The study of the petrographic features shows that there are several types of igneous rocks and two limestone formations in the Firmeza district. The igneous rocks form a continuous series from diabasic extrusive and intrusive rocks, through diorite and granite, to highly acid aplites. This series is cut by dike rocks of various types, which range in composition from basalt to andesite. The sedimentary rocks fall naturally into two divisions;—earlier marmorized limestone, probably Cretaceous in age, and now exposed only in scattered outcrops, and younger recent coral limestones deposited as a fringe on eroded igneous rocks.

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[16] *Op. cit.*, p. 12.

DISTRIBUTION OF THE VARIOUS ROCK TYPES

Surface Distribution

The two appended maps show the surface distribution of the rocks and orebodies. Owing to the lack of topographic maps, and the difficulties attending work in an area covered by tropical verdure, they cannot lay claim to great accuracy of detail. Future work will undoubtedly shift the contacts in many places and add a number of dikes. But enough work was done, and enough outcrops plotted to justify the making of the maps and the belief that they show the general relationship of the formations. Acknowledgment is made to Dean Corsa, whose unpublished map in the possession of the Juragua Iron Co. was in part used.

The different formations lie, as shown by the map, in belts of irregular width, roughly parallel to the coast line.

Nearest the sea is the belt of coral limestone. This belt is about 3,000 ft. wide in the area mapped. To the west, where the railroad runs to the Ocania Mine, the fringing limestone extends inland nearly 2 miles. Apparently the width of the belt is largely determined by the slope of the pre-deposition erosion surface, the wider belt lying ont he more gently sloping surface.

North of the coral limestone belt, the main mass of granitic rocks is exposed. This forms a belt from 6,000 to 8,000 ft. wide. Most of the granite area is covered by an alluvial deposit, through which rise knobs of the igneous rock. These knobs are all granite, usually with inclusions of dioritic material.

North of the granitic area lies the diorite. This forms a very irregular belt on the lower part of the foothills of the Sierra Maestra. From the Demajayabo River south of the Concordia Mine, to south of Loma Alta, the width of diorite is from 1,200 to 3,000 ft. At West Five Mine the diorite area runs back into the hills north of the region mapped. In the Juragua valley the diorite is exposed about 1 mile north of Firmeza.

The diabasic rocks lie higher in the hills than the diorite and in general their exposures on the surface are north of the diorite. No attempt has been made to show on the map the contact between the extrusive and intrusive facies of the diabases. This contact is exceedingly indefinite and rarely distinguishable.

Bodies of granitic rock, that differ from the main granite mass, cut through the diorite and the diabases in the region mapped. These bodies are apophyses of the main granite massif and form dikes, sills, or irregular bosses. They lie in a roughly east-west belt in the foothills, and are of great importance because the orebodies are scattered irregularly as a fringe to these apophyses. This can be seen clearly both on the map of the district and on the larger scale map of the mines near Firmeza.

The older limestones lie in the diorite and diabase area, rarely in contact with the granite. They have been metamorphosed to a dense marble in the area mapped, and are preserved as irregular masses capping the foothills, or as beds interbedded with the extrusive rocks. The outcrops vary in size from small blocks to areas that are as much as 1,200 ft. in diameter. An even larger area at the Ocania Mine does not show on the map.

All the formations are cut by the later basic dikes. Except in the mines these dikes have not been mapped. They fall into two systems: one almost vertical, striking northerly, or slightly east of north; the other an almost horizontal system, that might equally well be classed as a system of sills. Petrographically there is no difference between the two systems.

Vertical Distribution

The vertical arrangement of the different rock types is even more marked than their linear distribution, as it shows on the surface maps.

The coastal limestones rise in terraces to an elevation of about 300 ft., where they terminate in a flat top.

The granite massif has an upper surface which rises gradually from an elevation of about 350 ft. at the eastern end of the district, to about 700 ft. at the Ocania Mine. The granitic apophyses, as now exposed, are rarely more than 150 ft. above the top of the granite massif, and their downward extension where exposed by erosion can be seen to merge into the granite.

The diorite and the diabasic rocks must be considered together, because, while the lower limit of the diorite is determined by the top of the granite intrusion, the upper limit is, as a rule, indeterminate, and the diorite merges into the diabases. The two rocks were mapped separately in the field and a contact drawn, but it is generally arbitrary. More rarely the contact mapped represents an observed intrusive contact. As established, the contact between diorite and diabase rises from 500 ft. elevation in the eastern to 800 ft. at the Ocania end of the area. Fragmental igneous rocks are found at 700 ft. in the East Mine, and at 1,000 ft. elevation at Ocania. No continuous contact between intrusive and extrusive diabase was mapped.

The older limestone masses have not been found in this district, at a lower altitude than 400 ft. The highest body of limestone examined was the tuff-limestone found at 1,400 ft.

The lack of contour maps makes it difficult to draw accurate sections. But the elevations can be tabulated as follows:

West End of District (Ocania Mine), Feet		East End of District, Feet
1,000	Fragmental igneous rocks	700
800	Diorite-diabase contact	500
700	Top of granite massif	350
300	Top of coastal limestone	300

This table shows the distinct vertical distribution of the igneous rock types, and a well-defined pitch of their surfaces of contact.

Causes of Present Areal and Vertical Distribution of the Rock Types

Faulting.—The linear distribution of the rock types, as shown on surface maps, suggests at once a direction of major faulting. But except for two minor faults in the East Mine, the one northeast-southwest, the other northwest-southeast in strike, and a somewhat larger east-west fault in West Five Mine, no displacements worthy the name of fault were found in the area. The nearest approach to a fault system is shown by the later basic dikes. These, with the one vertical north-south trend and a horizontal system, appear to have filled a definite fissure system, but there is no evidence of any appreciable movement along most of them. In West Five Mine an aplitic dike is offset 6 in. on the opposite sides of a 2-ft. wide basic dike. If the dikes do represent an older fissure system, it is a system due to contraction during cooling of the igneous rock, rather than to faulting. The Firmeza district must be added to the long list of those in which ore deposits are connected with fissures of minor or no displacement.

It is the writer's opinion that the present areal and vertical distribution is due to three different causes—magmatic differentiation, tilting, and erosion. Each of three must be considered in an attempt to solve the problem.

Magmatic Differentiation.—To show that the vertical distribution of the rocks is due to magmatic differentiation it will be necessary to establish the comagmatic origin of the igneous rocks and their age relationship.

The detailed petrographic study has shown that there are present in the district igneous rocks showing all the stages of a gradual transition from diabase porphyry to highly acid aplites. The merging of the types can also be found in the field.

Northeast of Estancia hill the transition of a granite-porphyry sill into the main granite and quartz-diorite massif can be followed. There is no contact between the two types. The granite massif itself contains numerous inclusions of more basic material. These inclusions are angular, subangular, or rounded. J. F. Kemp[17] found apparently similar inclusions in the Daiquiri district and his interpretation is:

> "Apparently the inclusions represent some older solidified rock."

With this interpretation the writer does not feel in entire accord. The inclusions certainly represent early differentiates, either early crystal accumulations, or possibly segregations in the still liquid state which had an opportunity to crystallize; but either way they are more or less reabsorbed portions of the magma itself, and so endogenous inclusions.

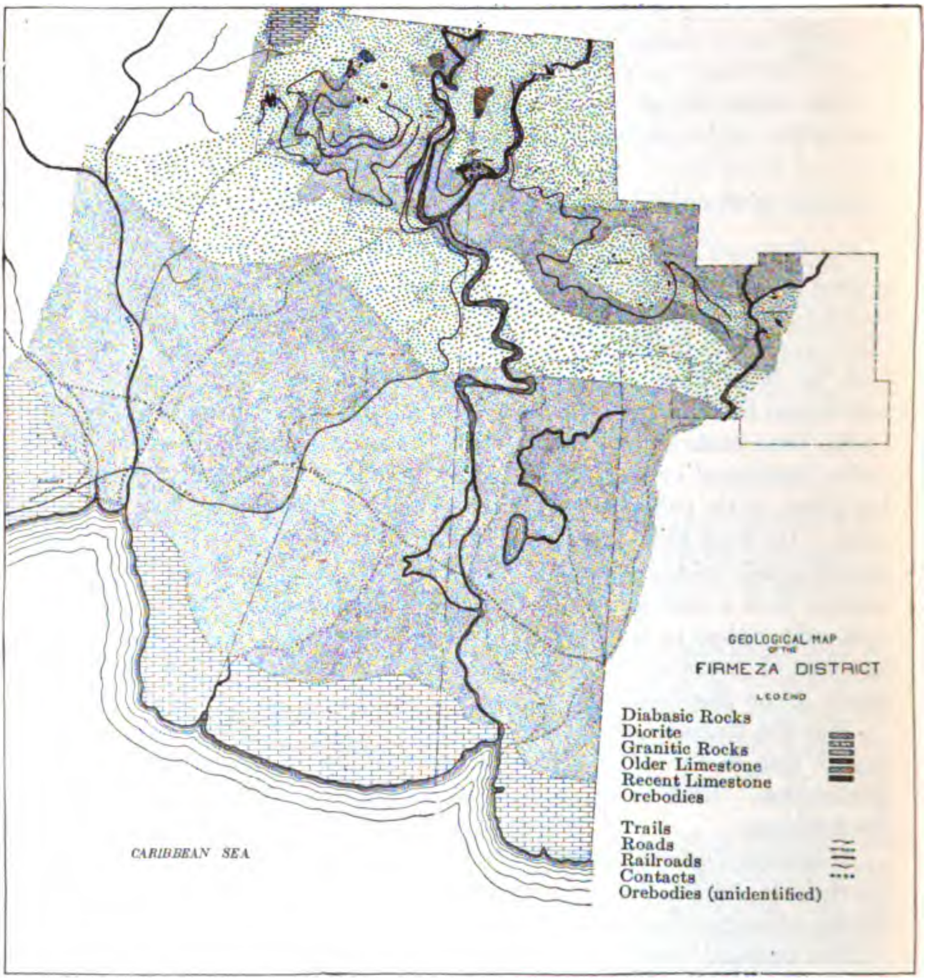

FIG. 9.

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[17] *Op. cit.*, p. 16.

After initiation of crystallization the character of the magma changed until it had power to reabsorb part of the already crystallized material. The result of the same action was observed on a small scale in a microphotograph of a specimen from the lower part of the aplite dike in West Five (not reproduced in this paper). It shows a zonal plagioclase, calcic in the central part, sodic at the edges, and much corroded. Obviously the plagioclase must have formed before the liquid was of such a composition that it could corrode it. Similarly it seems possible that, in a slow-cooling magma, crystals form and gather in clusters, and the interstitial liquid, changing in composition, could reabsorb them more or less completely.

Beside the evidence of the endogenous inclusions, the transition from quartz-bearing to quartz-free diorite can be observed in the upper levels of West Five Mine, and that from diorite to diabase in West Four Mine. That the extrusive diabases and intrusive diabases are of the same magmatic origin can be seen from the complete similarity of the two rocks both megascopically and microscopically. The only difference is the greater fineness of the ground mass in the volcanics.

Other features that seem to point toward comagmatic origin of the igneous rocks are the almost total absence of orthoclase in the entire series, and the predominance of hornblende as a ferromagnesian constituent. The latter is considered an indication of the presence of abundant crystallizers which would aid in holding the magma fluid at comparatively low temperature.

The age relationship of the igneous rocks in a country lacking in sediments that are chronologic guides must be determined by intrusive contacts. The intrusive nature of the granitic apophyses in the diabases and diorites can be established in almost any of the mines. That the main granite massif is later than the diorite is well shown in La Posa brook just north of its junction with the Rio Carpintero, where dikes of massive, even-grained granite cut the diorite. Diorite cutting diabase is seen at the contact of the two rocks in the Juragua valley north of the mines. Diabase containing fragments of the volcanic diabases is found all through the district.

The order of formation is, then, the normal one from the basic to the acid end of the series; *i.e.*:

1. Diabasic extrusive.
2. Diabasic intrusive.
3. Diorite.
4. Granite.
5. Aplite.

The origin from a common parent magma and the age relationship established, it can be shown how this would account for the observed

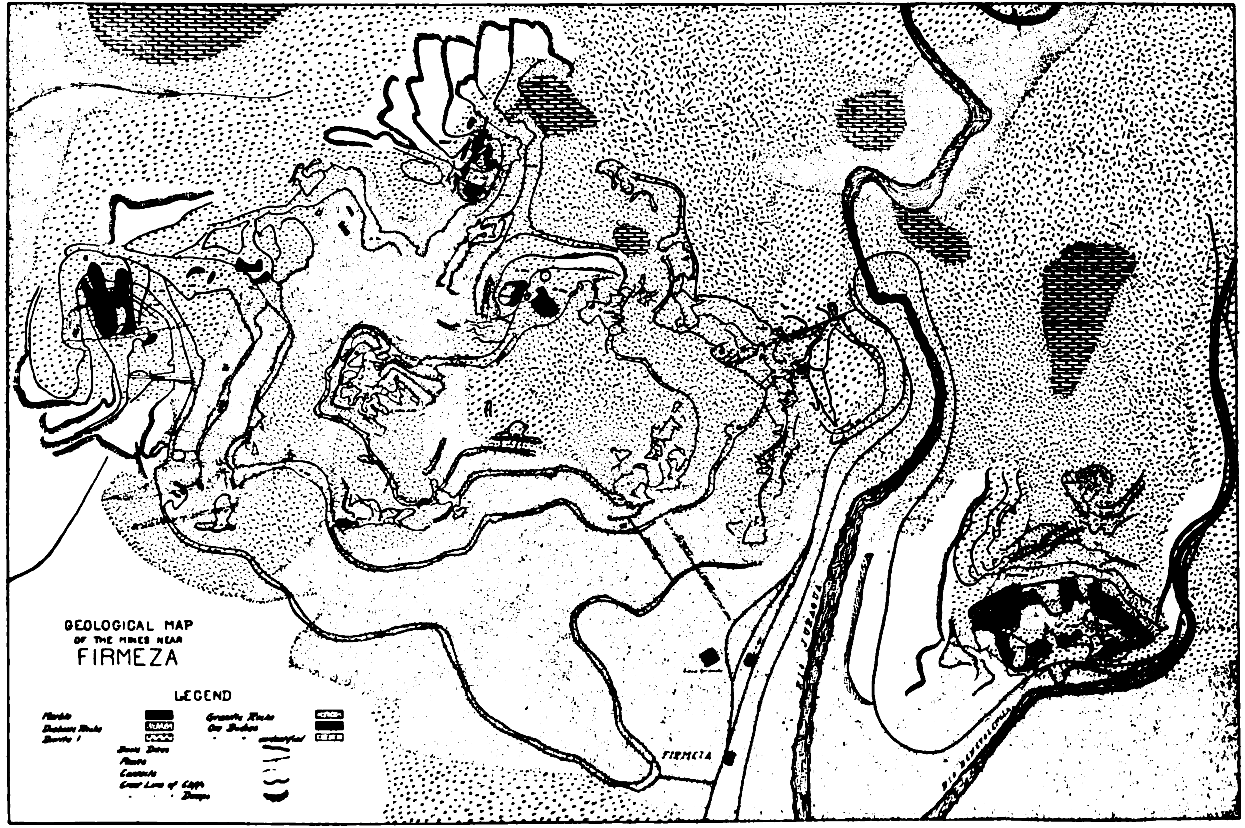
GEOLOGICAL MAP
OF THE MINES NEAR
FIRMEZA
LEGEND
Marble
Granite Rocks
Ore Bodies
Contacts
Crest Line of Cliffs
Dumps
FIRMEZA

Fig. 10.

vertical distribution of the igneous rock types. The series from diabase to granite agrees so completely with the series described by N. L. Bowen[18] as the normal result of fractional crystallization in a basaltic magma, that this mode of differentiation seems to be the probable one in this case.

The diabase extrusives mark the initiation of igneous activity, and the diabase intrusive and the diorite mark early chilled phases of a magma becoming more and more acid. The granitic massif is believed to be the batholythic invasion of the more completely differentiated acid liquid, and the granitic apophyses part of the final differentiate from the granite. They will be discussed more fully with the genesis of the ore deposits.

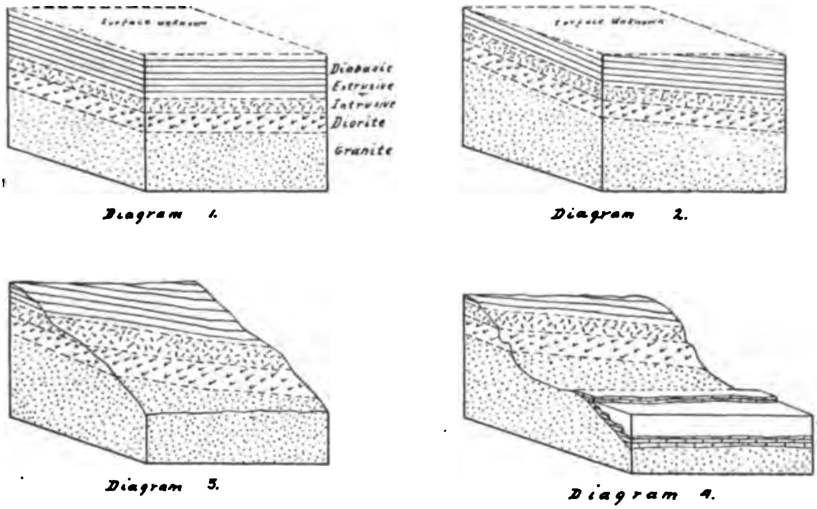

Fig. 11.—Genesis of Present Rock Types.

The position of the granite at a definite horizon is thought to be due to the fact that the cooled shell of the magma had developed cooling cracks to that depth, and as a result the crystallizers were able to escape from the granite, which could then no longer remain liquid.

Thus differentiation may account for the horizontal arrangement of the rock types observed in the field.

Tilting.—However, the arrangement is not entirely horizontal. There is a distinct slant toward the southeast of all the surfaces of contact. The extrusives and the limestones, where bedding can be determined, all show a pitch to the southeast. Just when the tilting took place is difficult to establish, but since it does not affect the coastal limestone it may be placed as previous to its deposition.

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[18] N. L. Bowen: The Later Stages of the Evolution of the Igneous Rocks. *Supplement to Journal of Geology*, Vol. 23 (November–December, 1915).

Whether tilting accompanied or followed the igneous cycle and the ore deposition cannot be determined. As the orebodies in the eastern end of the district are lower than those at the western end, it seems more probable that the tilting took place later than the period of ore deposition. The evidence is inconclusive and indefinite, but it is unimportant as the time of the tilting cannot affect any of the major hypotheses.

Erosion.—Since the time of ore deposition erosion has produced a surface that slopes down in general from the north to the south. In this way it has cut across the basic rocks and into the granitic massif.

Fig. 11 shows a diagrammatic representation of the way the present arrangement of the rock types is believed to have been brought about. The diagrams are not to scale, and are intended merely as an aid to visualizing the conditions.

Diagram 1. Shows conditions after differentiation and before tilting or erosion.
Diagram 2. Shows conditions after tilting and prior to erosion.
Diagram 3. Shows condition after partial erosion but prior to deposition of coastal limestones.
Diagram 4. Shows conditions as they are at present.

V. GENERAL DESCRIPTION OF ORE DEPOSITS

NATURE OF ORE

The ore mined in the Firmeza district is a mixture, in varying proportions, of magnetite and hematite. The ore is remarkably pure and contains little foreign matter. J. P. Kimball[19] gives the following figures:

	Per Cent.
Moisture (in part hygroscopic)	0.24 – 0.81
Silica and insoluble	5.00 –10.50
Phosphorus	0.009– 0.065
Sulphur	0.045– 0.248
Iron	61.00 –68.50

Although present mining methods make it possible to ship ores somewhat lower in iron content, and greater depth in mining has exposed ores somewhat higher in sulphur, the figures Kimball gave in 1884 are substantially correct for the ores now being mined. The low phosphorus and the absence of appreciable amounts of titanium make the ore a very valuable one for the manufacture of high-grade steel.

SHAPE AND SIZE OF OREBODIES

In shape the deposits are extremely irregular. Whatever their size they show one common feature throughout the area—a far greater extension in two dimensions than in the third dimension. They resemble a series of scattered lenses of irregular outline, that lie in every conceivable position. Fig. 12 shows a series of horizontal projections of the ore-bodies and a few vertical sections. The projections are drawn to scale, and the sections are sketched from exposures in the walls of the mines. The outlines represent the boundary of the ore of commercial grade. They do not represent the edge of the mineralization. There is generally a transition from ore to rock, not a definite contact between them. This subject will be more fully discussed under the mineralogy of the ore deposits.

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[19] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Science*, Ser. 3, vol. 28, p. 426 (1884).

The size of the ore deposits varies from pockets containing a few tons to lenses whose larger diameters are measured in hundreds of feet and whose thickness is from 10 to 50 ft.

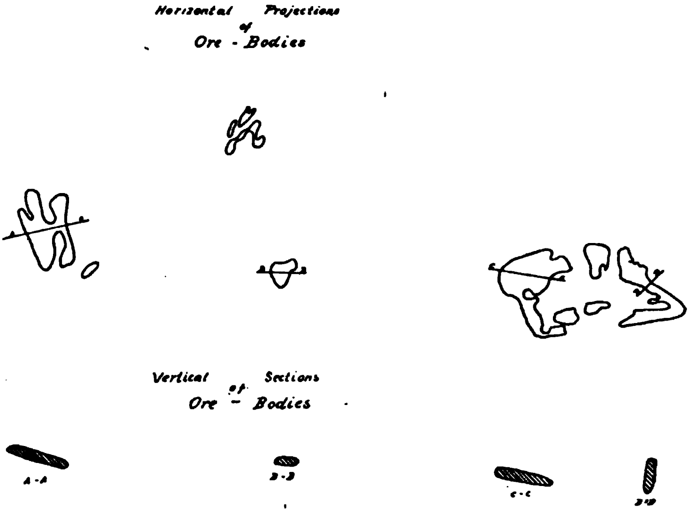
Horizontal Projections of Ore-Bodies
Vertical Sections of Ore-Bodies
A-A
B-B
C-C
D-D

FIG. 12.

GEOLOGIC POSITION

With the possible exception of the Ocania and Chicharron Mines, the ore deposits lie within 200 ft. of the granitic apophyses. These exceptions are probably more apparent than real and may indicate that the aplite has not yet been exposed, not that it is absent. In most cases the ores are directly in contact with aplitic or pegmatitic rock.

The relation of aplite to ore is exceedingly intimate. In most of the mines the ore lies against a floor or wall formed either of aplite or of pegmatitic granite with much micrographic quartz-plagioclase intergrowth. Frequent aplitic dikes traverse the ore, and form prominent features because of their light colors, in strong contrast to the black ore and the green igneous rocks. In spite of their apparent position as dikes, they are *not* to be considered as later than the mineralization. This will be shown in the discussion of the genesis of the ores.

The host rock of the ore is either one of the basic igneous rocks, or limestone. By far the greater part of the ore is in diorite or intrusive diabase; some is found in the diabasic extrusive rocks, and only a minor amount in the limestones. In at least three of the mines the ore deposits are very close to the overlying limestone masses, but there has been no localization in the limestone, and the larger masses are separated from the limestone by basic igneous rock.

MINERALOGY

The ore minerals are magnetite and specularite, with some amorphous, more or less hydrated, hematite in the upper parts of the ore deposits. The common gangue minerals are quartz, wollastonite, epidote, and lime-iron garnet. The epidote comes in two clearly distinct forms. The one is a well-developed, crystalline form, transparent and pleochroic from green to yellow. The other is a fibrous and scaly form, lacking in well-defined terminations. The well-crystallized form is the one that is associated with the ores, and other lime silicates. In some of the mines calcite is intimately intergrown with the ore. Apatite, titanite, and a mineral determined with some doubt as scapolite, were also found in rare cases. Secondary calcite, introduced by recent weathering, is not uncommon. Pyrite and chalcopyrite are found in most of the mines. They are almost always introduced and rarely if ever original constituents of the ore deposits. The pyrite occurs in well-developed crystals. One of these from the West One Mine is shown in Fig. 13. Associated with them is chlorite and some sericite.

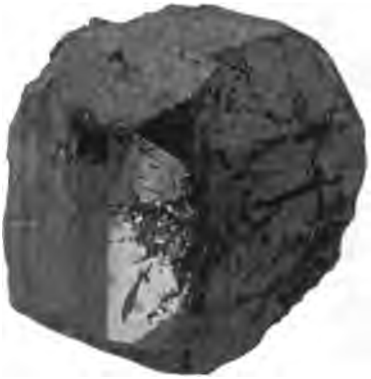
FIG. 13.—PYRITE CRYSTAL FROM WEST ONE MINE.

Distribution

The distribution of the minerals appears to be haphazard and extremely irregular, but natural phenomena are not haphazard, they are logical and sequential. Detailed study has convinced the writer that the minerals in these deposits are distributed according to a definite order. The most clearly localized material is the massive, fine-grained intergrowth of magnetite and specularite. This massive ore lies in, and close to, fissures, or else directly next to aplitic or pegmatitic granite. In the minute intergranular spaces of this ore occur wollastonite and some quartz. As the ore becomes less massive toward the margin of the deposit the proportion of specularite to magnetite increases, wollastonite becomes more abundant, as does quartz, and epidote and garnet appear. Still further out from the massive ore, garnet predominates, with some epidote, little or no wollastonite, very little quartz, and some more coarsely crystallized specularite and magnetite. In some cases this zone is composed entirely of massive garnet with very few impurities. Beyond the garnet is an area in which epidote and quartz are prevalent. Mineralization of this type is the most widely diffused, and merges gradually into partially chloritized and epidotized country rock. The above description holds only for the typical deposits in igneous rocks. The complete sequence is rarely well-exposed. It can best be observed in the East Mine and in West Five Mine.

Those deposits which are obviously in limestone show two kinds of mineral distribution. One is shown in the Chicharron Mine. It is a dike-like mass of magnetite and specularite, with interstitial wollastonite. The contact with the marble is sharp, and no contact minerals are found. The other form of orebody in limestone is a central mass of almost pure specularite, in rosette-like clusters, or granular masses, surrounded by an intimate intergrowth of quartz well-crystallized calcite, garnet and epidote. This form of orebody is best exposed in the North Mine. The transition to unaltered marble is not exposed there. In the Ocania Mine this transition is exposed and shows a sharp change from garnet rock to unaltered marble.

Apatite is so rare that it is a curiosity in the district. Fig. 14 shows a microphotograph of the one section in which it was found in appreciable amount. It appears in the usual hexagonal basal, and elongated prismatic sections. Later than the apatite in time of crystallization are magnetite and the minute garnet-epidote intergrowth. The scarcity of apatite and so of phosphorus in the ore is of the greatest economic importance. It is also an indication that phosphorus had little or no share in the mineralization.

The secondary calcite, the kaolin, and the amorphous and hydrated hematite, are found either very close to the surface or in channels to which meteoric waters have obviously had access. Pyrite is common in all the mines and lies close to the surface in an unaltered, well-crystallized condition.

Interpretation of the Mineralogy

The first inference to be drawn from the mineralogy of the ore deposits is that they were formed at a high temperature. The temperature of mineral formations is not yet so well known that accurate conclusions can be drawn from mineralogy as to the exact temperature at which ore deposits were formed, but the occurrence of lime-iron garnet and of wollastonite indicates high temperatures. The association with quartzose pegmatitic dikes, which, as will appear later, are believed to be contemporaneous with the ore formation, places the temperature at from 575° to 800°C.[20]

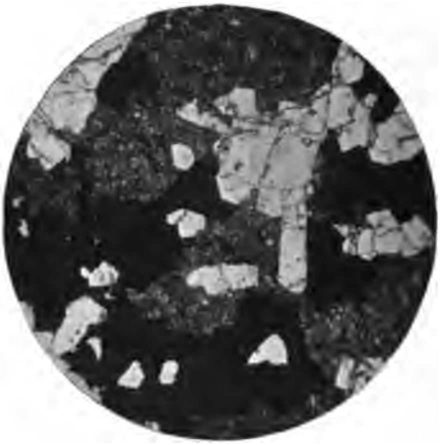

Fig. 14.—Specimen from West Five Mine. × 18. Magnetite—black. Epidote and garnet—gray. Apatite—clear white.

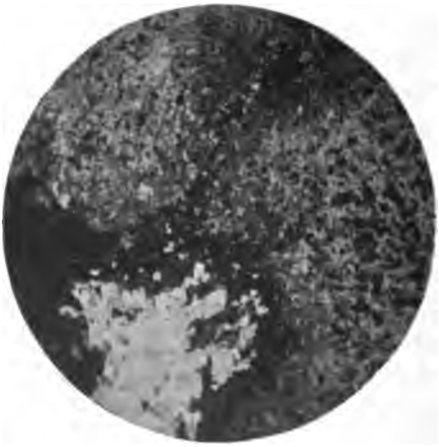

Fig. 15.—Diabase Porphyry, Showing Introduced Quartz and Epidotization. From East Mine. × 18. Quartz—white. Epidote—dark.

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[20] F. E. Wright and E. S. Larsen: Quartz as a Geologic Thermometer. *American Journal of Science*, Ser. 4, vol. 27, p. 421 (1909).

The second inference drawn from the mineralogy is that the iron, lime, and quartz are either entirely, or in part, introduced material. The large quantity and great concentration of the iron oxides makes their formation from the rock in which they are found highly improbable, and it can be assumed that they represent, almost entirely, introduced material.

The lime silicates, epidote, wollastonite, and garnet are so abundant that an addition of lime seems extremely probable. Because the rock, now entirely replaced by ore, was a calcic igneous rock, it is impossible to determine whether the lime, which was added to form garnet zones, came from the magma or from the replaced rock. Since, in places like Estancia Hill, the garnet zones are out of all proportion to the size of the orebodies, an addition of lime from the magma seems probable. The major part of the lime is deposited in a zone beyond the iron. In this section the lime silicates always show a later crystallization than the iron oxides. This makes it appear as if the mineralizing agent deposited most of the iron oxides while still able to retain much lime in solution. Further evidence of the addition of lime from the magma is afforded by the apatite, intergrown with the magnetite, in the specimen from West Five Mine described above (Fig. 14). The epidotization and silicification in the outermost zone seems to be largely recrystallization, and represents a hydrothermal alteration, more or less *in situ,* rather than an addition of material. The chloritization is also an effect of this hydrothermal metamorphism. Six samples of diabase showing more or less epidote were analyzed for lime. They were selected at various distances from a limestone inclusion, in order to show a supposed absorption of lime. The variation in the lime content is so slight that no addition of lime in this zone can be postulated. The quartz is probably largely a byproduct of the alteration of hornblende to epidote. It is also in part introduced, as is shown in the microphotograph of a specimen from the diabasic porphyry in the East Mine (Fig. 15).

In regard to the chemical form of the mineralization, inferences are largely negative. Evidence of the participation in the mineralization of any of the halogens is practically lacking. The apatite and dubious scapolite indicate some chlorine, but the occurrence of chlorine-bearing minerals is so rare that the introduction of the iron as chloride cannot be shown. CO_2 would be expected to show its presence in the formation of carbonates in the zone of hydrothermal alteration. No appreciable amount of carbonates was found, except in the occurrences in limestone. The presence of much water is shown, however, by the extensive hydrothermal alteration in the outer zone of mineralization. If the inferences in regard to the temperature of the formation of the ore deposits is correct, it is more than 200° above the critical temperature of water (358°C.). The fact that this water carried material in solution makes it impossible to prove that it was in the form of vapor; but the high temperature and the fact that the mineralizers were able to permeate dense rocks so readily makes it probable that they were in the gaseous rather than the liquid phase. The mineralizers are, therefore, believed to have been dominantly water vapor, carrying with it iron, lime and silica. The variation in mineralization with decreasing intensity is believed to be as shown in the diagram, Fig. 16.

A later mineralization is represented by the pyrite and chalcopyrite. Field observation shows that the pyrite in the ore lies in channels along which solutions have passed. In the ore it is usually accompanied by chlorite. Where the pyritization has taken place in the wall rocks, as at the west side of the Ocania Mine, both chlorite and sericite occur. The pyrite seems to represent a very much later stage of mineralization than the magnetite and hematite. It is impossible to prove any definite connection between the pyritization and the mineralization that produced the orebodies. It seems probable that the solutions that brought the pyrite came in at the time of the intrusion of the later dikes.

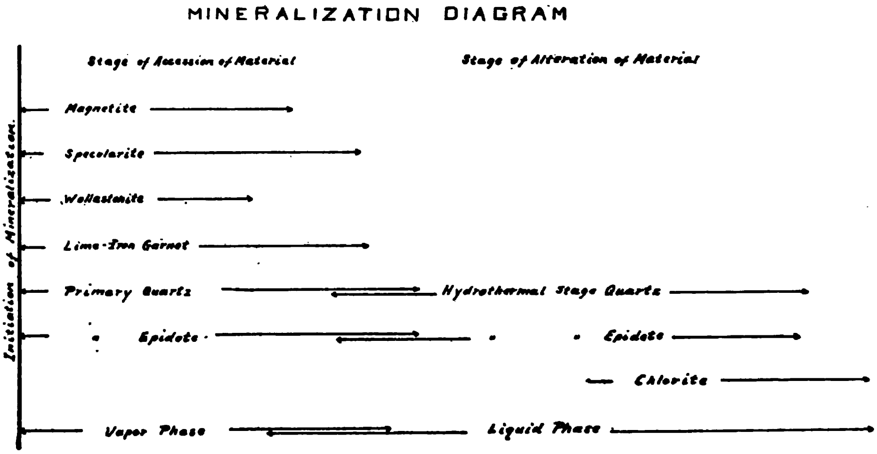

FIG. 16.

Superficial Alteration

W. Lindgren believes that the hematite is the result of the alteration of magnetite by surface agencies. He says:

> "——the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters."[21]

The occurrences in the Firmeza district are in entire accord with those recorded by Lindgren at Daiquiri. There is an increase of hematite in the higher levels of the mines. A polished specimen of the massive ore from the Chicharron Mine shows very clearly the intimate intergrowth of specularite and magnetite, and some of the specularite forms a later veinlet through the magnetite; but to accept without question specularite as an oxidation product of magnetite is difficult. The writer has been unable to find any record of the synthesis of specularite except at high temperatures, or of the occurrence of specularite as the result of surface oxidation except under superimposed regional metamorphism. It is not the crystalline facies of hematite that forms under surface agencies, as the result of dehydration of limonite, in the ores of Mayari. Lindgren does not state that the process of hematitization produces the crystalline facies of hematite (specularite). But since it is the crystalline phase that is abundant, and the earthy phase rare, it must be the specularite to which he refers. He does say:[22]

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[21] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vlo. 53, p. 52 (1916).

> "The pits in the hematite are probably caused by the local development of a softer or earthy facies of the mineral."

In spite of the fact that all the observations accord with the theory advanced by Lindgren, the writer desires to suggest another hypothesis—that is, that both the magnetitization and the hematitization are due to a primary mineralization by gaseous solutions deficient in ferrous oxide.

F. W. Clarke says:[23]

> "Ferric oxide can crystallize out as hematite only when ferrous compounds are either absent or present in quite subordinate amounts, for ferrous oxide unites with it to form magnetite."

The crystallization of specularite as a later phase, from a gaseous solution containing insufficient ferrous oxide to produce all magnetite, is in accord with Clarke's observation. Deficiency of FeO could account for a change from magnetite to specularite about centers of crystallization. It could also account for a diffusion of the hematitization to a higher level, further from the centers of mineralization than the magnetitization. Local variations in the mineralizing solutions account for the variations in the proportion of hematite to magnetite in bodies equally close to the parent magma.

The only important effect ascribed to surface waters by the writer is the oxidation of the pyrite near the surface, and the resulting decrease in the undesirable sulphur content.

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[22] *Op. cit.*, p. 52.

[23] F. W. Clarke: Data of Geochemistry. *Bulletin* No. 616, *U. S. Geological Survey*, p. 347.

VI. GENESIS OF THE ORE DEPOSITS

PREVIOUS THEORIES

The preceding description of the ore deposits, and discussion of their mineralogy, makes it possible to consider the larger features of their mode of formation. As this is purely a matter of interpretation, it is necessary to consider first the interpretation given by others. J. P. Kimball, writing in 1884–85, and F. F. Chisholm, in 1890, formed their conclusions from observations made in the Firmeza district. A. C. Spencer based his theories on work done at both Firmeza and Daiquiri in 1901. The papers by J. F. Kemp, and by W. Lindgren and C. P. Ross, were written more largely from evidence gathered at Daiquiri, with some specimens and a brief visit by Kemp to the Juragua mines in 1914. J. T. Singewald and B. LeRoy Miller made a brief visit to the Firmeza and Daiquiri districts in the fall of 1915.

J. P. Kimball[24] recognizes two types of ore deposit in the Juragua district. These are "replacements" of the coral limestone by iron-bearing solutions, and "concentrations" of ferric oxide in the diorite, "almost *in situ*." Both replacement and concentration are ascribed to the action of circulating meteoric waters.

F. F. Chisholm[25] discusses the theory advanced by J. P. Kimball, and disagrees with it. He states his own opinion as follows:[26]

> "My conclusions, after going in detail over most of the exposure made by the Union cut, were that, whatever the exact character of the ore deposit, the present position of the ore cannot properly be considered the result of local metamorphism of limestones by the action of surface waters containing iron leached from the overlying mass of iron-bearing diorite. I am much more strongly inclined to consider the ore here either the result of concentration within a diorite dyke which was originally characterized by the presence of a large percentage of iron, or else a distinct band forming a portion of a larger dyke. In other words, I am strongly of the opinion that the source of the ore is from below, and consequently that the loyalty of these deposits may be relied on below the limits of atmospheric action. I regret that I was unable to go into the question in detail, and get positive facts in support of my belief, but that I am obliged to admit that my examination was too superficial to enable me to prove my views."

A. C. Spencer,[27] after considering several possible explanations of the genesis of the ore deposits, comes to the conclusion that they are parts of an older limestone-schist series which has been completely involved in the later igneous intrusions. The following quotation shows how completely Spencer supposed this older series to have been immersed in the igneous rock.[28]

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[24] J. P. Kimball: Geological Relations and Genesis of the Specular Iron Ores of Santiago de Cuba. *American Journal of Sciences*, Ser. 3, vol. 28, p. 426 (1884).

The Iron-Ore Range of the Santiago District of Cuba. *Trans.*, vol. 13., pp. 613–634 (1884–85).

[25] F. F. Chisholm: Iron-Ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. 3, part 3, pp. 259–263 (1888).

[26] *Op. cit.*, p. 262.

[27] C. W. Hayes, T. W. Vaughan, and A. C. Spencer: *Report on a Geological Reconnaissance of Cuba*, 1901.

> "A mass of many million tons weight floated upward by the buoyant effect of molten rock in motion from the interior toward the surface of the earth, is the only conception which adequately accounts for the mode of occurrence of the orebodies of the Magdalena and the Lola Mines at Daiquiri; while, though less strikingly shown, at Firmeza it is likely that the masses of schist, marble and ore have been likewise actually suspended in the molten lava."

J. F. Kemp[29] divides the orebodies into two types—the "Distinctive Contact Zones" in limestone, and the orebodies in the diorite. The former he regards as due to contact-metamorphic effects produced in the limestones by the granite. Of the orebodies in the diorite he says:

> "One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or srom included blocks of limestone, or deep-lying limestone, or from several of these fources."

Another quotation from the same article might indicate that Kemp suspected what has become the conviction of the writer:[30]

> "One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it."

Kemp apparently bases his conclusion that the aplite dikes represent a later intrusion into the ore on the fact that the dikes appear to cut the ore. That another hypothesis is possible will be shown later.

W. Lindgren and C. P. Ross,[31] on evidence gathered by the senior author, deduced the following conclusions:[32]

> "From the above it is clear that the primary iron oxide of the deposits at Daiquiri is a magnetite, which subsequently has been altered more or less completely to a

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[28] *Op. cit.*, p. 82.

[29] J. F. Kemp: The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. *Trans.*, vol. 53, p. 30 (1916).

[30] *Op. cit.*, p. 21.

[31] W. Lindgren and C. P. Ross: The Iron Deposits of Daiquiri, Cuba. *Trans.*, vol. 53, p. 52 (1916).

[32] *Op. cit.*, p. 53.

> hematite. The discussion of genesis may therefore be divided into two parts: the first relating to the origin of the magnetite; the second to its subsequent hematitization.
>
> The mineral association of the magnetite, particularly the presence of much garnet, shows that it originated under high-temperature conditions.
>
> On the other hand, the hematitization is probably a low-temperature process developing gradually under the influence of oxidizing atmospheric waters."

The problem of hematitization has already been discussed.

Further on three theories discussed by A. C. Spencer are rediscussed by the authors. After dismissing the other two theories, the second theory is accepted, as follows:

> "There remains the second theory, accounting for the large masses of magnetite included in the diorite by contact-metamorphic limestone, by which the latter has become almost entirely replaced by magnetite derived from magmatic emanations rich in iron. While it is freely admitted that the genesis of the Daiquiri deposits is difficult to explain, it will be shown that the view outlined in the previous sentence has much in its favor."

J. T. Singewald and B. LeRoy Miller discuss the theories advanced by J. F. Kemp, and by W. Lindgren and C. P. Ross in the papers cited above and come to the conclusions summed up in the following paragraph:[33]

> "To sum up our opinions, the Cuban iron ores are contact-metamorphic deposits localized about engulfed blocks of limestone in diorite. In such cases, where there was a limited supply of magmatic emissions, there resulted the contact metamorphism of only a part of the limestone block. Where the supply was ample and the action most intense, not only was the block of limestone completely replaced, but complete endomorphism of the igneous rock on a large scale occurred in the vicinity."

The above are the interpretations offered by others who have studied the field more or less closely. But a study of different exposures and a different viewpoint, have led the writer to conclusions which are somewhat at variance with those quoted.

HYPOTHESIS OF THE GENESIS OF THE ORE DEPOSITS

Any comprehensive theory concerning the formation of an ore deposit in a rock of which it is not an original part—*i.e.*, an epigenetic deposit—must account for four things:

I. The source of the material forming the deposit.
II. The vehicle that introduced the material.
III. The channel through which the material was introduced.
IV. The cause for the deposition of the material.

Source of the Iron Ore

The ultimate source of the iron ore was the diabasic magma. The extruded part of this magma, with the involved limestones, together with the chilled upper part of the magma (the diabases and diorite) form the host rock of the ore deposits. It has already been shown, in the discussion of the areal geology of the district, that this magma was differentiated, that the earlier igneous rocks represent the parent magma and its early differentiation products, and that the latest rock differentiate was the granite massif with its aplitic apophyses. These aplites, which are, under anhydrous conditions, extremely viscous, must have contained large quantities of crystallizers, to keep them fluid and enable them to penetrate the basic rocks as sills and dikes. It is believed that these crystallizers were mostly composed of water vapor well above the critical temperature, and that they held in an ionized state the ore minerals.

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[33] J. T. Singewald and B. LeRoy Miller: The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba. *Trans.*, vol. 53, p. 73 (1916).

Vehicle to Carry the Material

These concentrated crystallizers were also the vehicle to carry the mineral burden into the rocks which act as host for the ore deposits, *i.e.*, the crystallizers plus the minerals are the mineralizers. Their nature has already been discussed under the interpretation of the mineralogy.

Channels through which the Materials Were Introduced

One of the striking features of the larger Firmeza deposits is that the ores have almost obviously avoided the masses of marmorized limestone. This can be explained by the fact that the channels, through which the mineralizers entered, were not lines of weakness or shear zones produced by structural faulting, but were cooling cracks in the igneous rocks.

Causes of Deposition

The causes of deposition may be of either a chemical or physical nature, or a combination of the two. It has been the tendency of geologists to ascribe to chemical action the dominant rôle in contact-metamorphic actions. This makes it difficult to account for ore deposits in two rocks as chemically different as diabasic igneous rock and marmorized limestone, except by postulating different theories. If it is believed that the dominating cause of deposition is physical, and is inherent in the mineralizing solutions, whether liquid or gaseous, this difficulty is obviated. It is the writer's opinion that such was the case in the formation of these deposits. The mineralizers were under such great pressure and at such high temperature that they had the power to diffuse through, and either partially or wholly absorb, any rock with which they came into contact. This diffusion and partial absorption would rob the mineralizers of a large part of their activity, and would cause deposition of the minerals in a definite order. The order would be dependent on the saturation of the gases or solutions with the material to be deposited. The field evidence for such an order is clear, and as shown in discussing the mineralogy of the deposits, magnetite and specularite are the first minerals deposited. As a result, the magnetite and specularite are localized in and near the fissures through which the solutions had access, while the epidote, which is probably only in part due to accession of material from the magma, and is more largely a recrystallization of material present in the host rock, is much more widely diffused. Joseph Barrell, in conversation with the writer, suggested that such a hypothesis may also account for the association of aplitic and pegmatitic dikes with basic mineral segregations. The dikes are regarded as the residual, acidic rock material left in fissures, from which the mineralizers had gone into the rockbody, and in the process had effected a separation of acidic and basic material.

Under the hypothesis advanced, the orebodies owe their localization in igneous rock or limestone almost entirely to the accessibility of the rocks to the mineralizing solutions. The same causes govern all the cases.

Briefly stated, it is the writer's belief that the orebodies in the Firmeza district are due to mineralizers concentrated by further differentiation from the granitic massif, which is itself a differentiate of the diabasic magma. These mineralizers diffused from the igneous-rock material into those rocks to which fissures gave access, and, in diffusing, deposited their mineral burden, and left the rock material with which they had been mixed in the fissures, as aplites and pegmatites.

Under this hypothesis the mineralization becomes a definite event in the igneous cycle, which cycle may be restated as follows:

1. Extrusion of diabasic rock material.
2. Intrusion of diabasic magma.
3. Formation by differentiation of more acidic residual liquid in magma reservoir, with concentration of mineralizers.
4. Batholythic invasion of granitic material, together with mineralizers.
5. Further differentiation of the granitic batholith and escape of mineralizers with some acidic rock material, into cooling cracks in country rock.
6. Separation of basic mineralizers, and acidic rock material, with contemporaneous formation of ore deposits and aplitic dikes and pegmatitic apophyses.

The hypothesis submitted above is given prior to a detailed description of the ore deposits. It must be tested by its application to the phenomena observed in the field, and by its ability to meet the objections raised by other writers on the same subject.

DISCUSSION OF THE DIFFERENT HYPOTHESES

The hypotheses of the different geologists who have visited the region have been given, either as abstracts or by quotations. The earliest of these, that by J. P. Kimball, does not account for the formation of what are now known to be high-temperature minerals: garnet, wollastonite, etc. Kimball recognized, however, that some of the ores lay in diorite and could not be accounted for by a hypothesis which depended entirely on limestone as a precipitant. F. F. Chisholm does not attempt to formulate a complete theory of the genesis of the ore deposits. With his conclusion that the mineralizing solutions came from below, and were not of meteoric origin, the writer entirely agrees.

The theory accepted by A. Spencer requires an older limestone-schist series. Limestone was found in the Firmeza district in considerable abundance, but schist could not be found by the writer. The occurrence of tuff-limestones interbedded with the volcanic extrusive material makes it seem probable that the period of sedimentation, and the initiation of the igneous cycle, were not separated by a time interval sufficient for the forming of surface orebodies and their entombing in igneous rock. This theory also fails to account for the forming of ore in diorite. In a tunnel under West Five Mine the transition from fresh diorite to granular magnetite can be traced. The transition is gradual and there is no contact which could possibly be interpreted as the margin of a partly assimilated block.

Later writers all agree in ascribing an igneous origin to the mineralizing solutions. The principal divergence is in the amount of influence ascribed to the limestone. Lindgren and Ross believe the ore to be definitely localized in limestone. Singewald and Miller regard the ore as in, and about limestone. Kemp believes that some of the deposits are independent of limestone. In regard to this feature the writer agrees entirely with Kemp. The conception of the orebodies as metamorphosed, engulfed limestone masses is difficult. The shape and position of the ore-masses demand that the entombed blocks should have been comparatively thin, and that they should have come to rest in every conceivable position. This theory completely fails to account for the gradual transition from ore to diorite shown at its best in West Five Mine. Such a deposit could be accounted for only on the basis of complete assimilation of the limestone, and a deposition of magnetite about the locus of assimilation. Any theory that places the burden of producing the orebodies upon the diorite, is open to one great objection. It demands that a magma of almost gabbroic basicity shall have produced contact-metamorphic phenomena of the most intense variety. Kemp recognized this, and ascribes the orebodies in limestone to the granite, and the orebodies in the diorite to solutions from "some other" intrusive. But Lindgren and

Ross, and Singewald and Miller regard the diorite as the source and as the cause of the mineralization. Another objection is the fact that, under that theory, the diorite must have exercised a strong selective tendency. Unaltered marble within 20 ft. of massive ore, and separated from the ore by recognizable diorite, can be seen at West Four. Such a case is not rare, it is a common feature. In one of the old cuts above West One Mine there is exposed a small block of marble, completely surrounded by diabase containing small ore masses. But the marble is entirely unaffected. Contact-metamorphic phenomena are undoubtedly capricious, but if another explanation will solve the problem in such a way as to show that the capriciousness is more apparent than real, it must at least be considered.

The great objection to ascribing the source of the ores to the granitic batholith lies in the interpretation of the aplitic and pegmatitic dikes. Field observation clearly establishes the direct connection between these rock types and the granitic massif. If these most acidic facies represent a later intrusion into the ore, the ore cannot come from the granite. Kemp believes they are later. He says:[34]

> "——whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies."

Lindgren and Ross[35] agree with this.

> "The dikes of igneous rock[35] intruded into the iron ore appear to be somewhat different from the diorite and are either granite porphyry or aplite; that is, complementary dikes of a later generation."

F. Klockmann, writing on contact-metamorphic magnetite deposits in general, puts the matter even more strongly:[36]

> "Wiederholt wird angegeben, dass aplitische Gänge und Granitapophysen die Magnetitlager stätte durchsetzen. Gibt es denn für den Unbefangenen dafür noch andere Deutung als die, dass der Granit in eine von ihm vorgefundene Lagerstätte eine Apophysen entsendet hat?"

As a general proposition, the very fact that aplitic dikes and granite apophyses are repeatedly reported as traversing magnetite deposits is the strongest possible argument against the purely fortuitous intrusive nature of such apophyses. It was this constant relationship observed in the field study, that first convinced the writer that there must be definite connection between the forming of the orebodies and of the granitic apophyses.

J. H. L. Vogt recognizes a connection between the granite and its apophyses, and the magnetite deposits of Kristiania. He says:[37]

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[34] *Op. cit.*, p. 21.

[35] *Op. cit.*, p. 54.

[36] F. Klockmann: Über Kontaktmetamorphe Magnetitlagerstätten. *Zeitschrift für Praktische Geologie*, vol. 12, p. 81 (1904).

[37] J. H. L. Vogt: Problems in the Geology of Ore Deposits. *Trans.*, vol. 31, p. 138 (1901).

> "A study of the Kristiania contact-deposits indicates that the formation of the ores preceded the solidification of the granitic magma. Even when the ores occur in slates immediately adjacent to the granite, or in the small Silurian fragments completely surrounded by granite, they are never found also in the granite itself. This is to be simply explained by the supposition that from the still liquid magma the ores were 'blown into' the adjoining rigid rocks. —The presence in these deposits of granitic apophyses, already mentioned, is another proof that they were formed before the solidification of the granite."

He regards the occurrence of the apophyses with the ore in the Kristiania district as proof that the granitic magma had not yet solidified.

The absence of ilmenite in the Firmeza occurrence is also an indication of their intimate connection with the granitic magma. Vogt has given some very complete studies of the relationship between the different magnetite deposits and their sources. A quotation that shows some of his conclusions follows:[38]

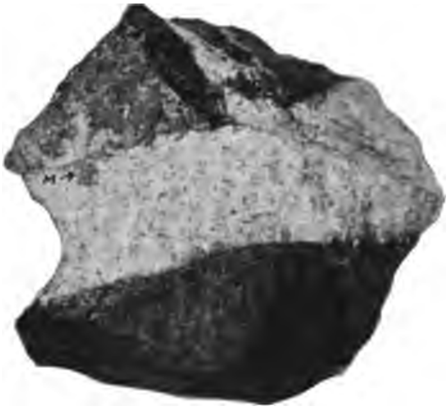

Fig. 17.—Aplitic Granite Bordered by Magnetite and Epidote. Tunnel at Elevation of 299 Ft. under West Five Workings. Magnetite Caught in Aplite = M. Actual Size. Aplite—white. Magnetite—dark.

> "Die Erfahrung ergiebt, dass bei der 'oxydischen' Erzaussonderung es namentlich das 'Eisenoxyd-mineral'—bei den Gabbros Titaneisen oder Titanomagnetit, bei den Peridotiten Chromit—ist, welches concentrirt wird. Es ist somit a priori anzunehmen, dass etwaige 'oxydische' Erzausscheidungen in den Graniten einen ähnlichen niedrigen TiO_2-Gehalt führen müssen wie die in dem Granit normal ausgeschiedenen (wohl namentlich Magnetit und Eisenglanz, untergeordnet Titaneisen.)"

The strongest evidence of the intimate relation between the granitic apophyses and the mineralization is based on field observation. Some of the small dikes are reproductions in miniature of the larger occurrences. Figs. 17, 18 and 19 are photographs of two of these small dikes.

Fig. 17 is of a specimen from the ore in the tunnel under West Five

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[38] J. H. L. Vogt: Zur Classification der Erzvorkommen. *Zeitschrift für Praktische Geologie*, vol. 2, p. 394 (October, 1894).

Mine. This is as deep-seated an ore deposit as has been exposed, and is the one most clearly independent of limestone. It is noticeable how entirely clear-cut is the contact between aplite and magnetite in the lower part of the picture. At the same time the other contact is less sharply defined and clearly shows some magnetite trapped in the aplite.

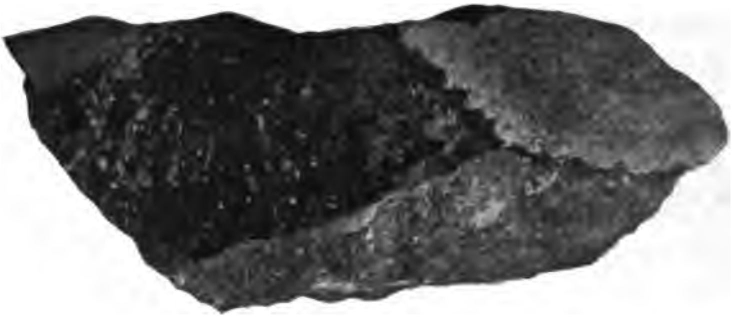


FIG. 18.—PHOTOGRAPH OF SPECIMEN FROM EAST MINE, SHOWING RIM OF GARNETS DEVELOPED BETWEEN APLITIC GRANITE (WHITE) AND DIABASE PORPHYRY (DARK). ACTUAL SIZE.

Fig. 18 is a photograph and Fig. 19 a microphotograph of a specimen from the East Mine. This specimen comes from the area of diabasic rocks, that lies between the ore in the East Mine and the ore in the pit above, known as North Mine. The specimen shows garnet, merging on one side into diabase porphyry, on the other into aplite. The writer can see only one explanation for this specimen, and that is a change in the mineralizing solutions from intermixed mineralizers and acidic rock material to aplite.

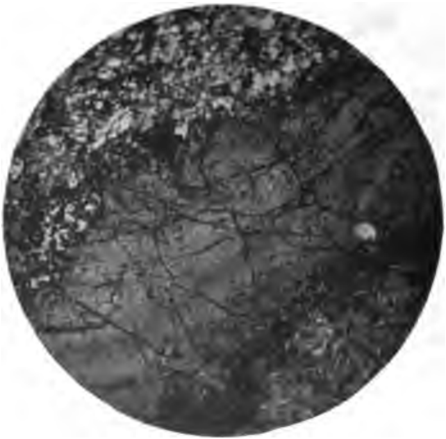


FIG. 19.—GARNET BAND BETWEEN APLITIC GRANITE (LOWER RIGHT) AND DIABASE PORPHYRY (UPPER LEFT). FROM EAST MINE. × 18.

It is on the basis of such occurrences, and for the reasons given, that the writer believes: that the aplite and ore are phenomena produced at approximately the same time; that both were formed from a mixture of mineralizers and rock material derived from the granitic magma; and that the iron oxides, lime and the major part of the silica, held in aqueous solutions above the critical temperature, diffused into the wall rock, leaving the residual rock material in the fissures as aplitic dikes.

VII. DETAILED DESCRIPTION OF THE ORE DEPOSITS

In order to test the hypothesis stated above in regard to the genesis of the ore deposits, it must be applied to the different occurrences of ore in the district. To do this, a detailed description of the more carefully studied mines will be given, and the mode of formation of the orebodies discussed.

OCANIA MINE

The Ocania is the most westerly of the Juragua Iron Co.'s mines. It lies on a steep hillside, at an elevation of from 860 to 1,085 ft.

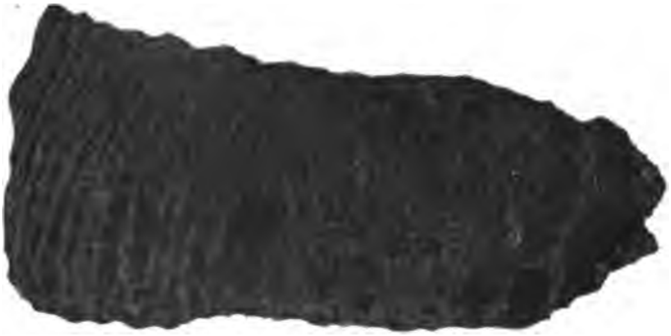


FIG. 20.—SPECIMEN FROM OCANIA MINE, LEVEL 2. THE DARK MATERIAL IS GARNET. THE LIGHT MATERIAL IS WOLLASTONITE. THE STRUCTURE IS ATTRIBUTED TO DIFFUSION. ACTUAL SIZE.

The wall rocks are coarse, diabase porphyry, volcanic agglomerates, and marble. The contact between the granitic and diabasic rocks lies on the south slope of the hill, at an elevation of about 700 ft. A quartz-bearing, andesitic dike is found in the mine itself. The ore forms a series of lenses that strike in a northwesterly direction, and have a steeply inclined dip to the southwest. These lenses are made up almost entirely of hematite and quartz, with little or no magnetite. There is considerable pyrite through the ore in various places, and, at the edge of the limestone, chalcopyrite and manganese oxide are found. Some fragments of diabase, with native copper in them, were observed near the limestone. Between the ore and the limestone are masses of garnet rock. Near the bottom of the orebody, as exposed on the southeast side of the mine, was found a rock (Fig. 20) which shows, in part, alternating layers of brown garnet, and a greenish mineral which the microscope shows to be wollastonite. As the accompanying microphotograph (Fig. 21) of the contact between the garnet and wollastonite shows, the two bands have an interlocking structure, perpendicular to the boundary. The black crystals in the wollastonite band are magnetite. The garnet does not show crystal edges. Toward the center it is very dusty, with undeterminable inclusions. The wollastonite forms an aggregate of poorly developed, prismatic crystals. A somewhat similar structure, made up of magnetite and actinolite, is shown in a specimen from the Juragua Mines described by W. Lindgren. He says:

> "The texture of this specimen strongly suggests diffusion banding, such as might form in a material, a limestone perhaps, freely penetrated by hot iron-bearing solution."

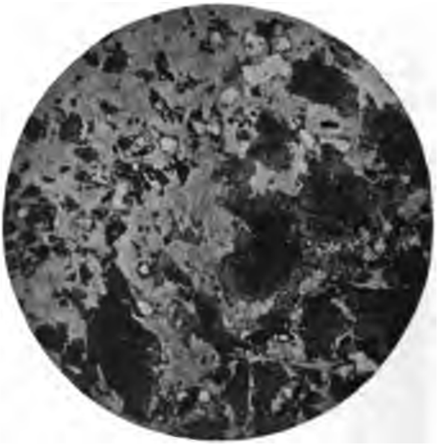


FIG. 21.—CONTACT BETWEEN GARNET AND WOLLASTONITE IN ROCK SHOWING DIFFUSION STRUCTURE. FROM OCANIA MINE. × 18. Magnetite—black. Garnet—dark. Wollastonite—grey. Holes in thin section—white.

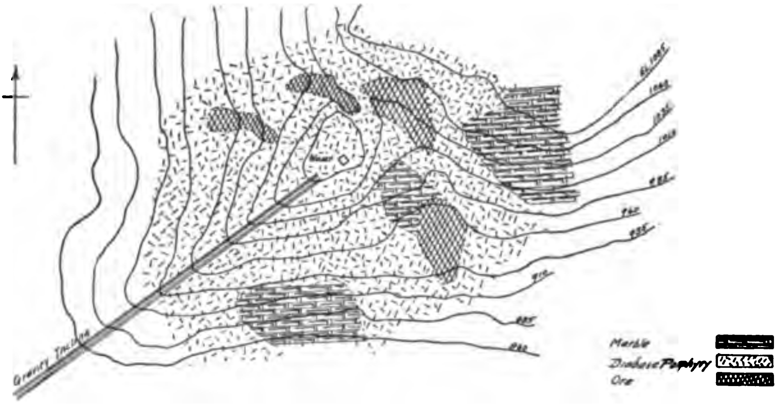
In the case of the Ocania specimen the solutions were probably lime-bearing and lower in iron. Whether the original rock was limestone or diabase porphyry, it is not possible to tell. It is true of this mine that the mineralization effects have spread further into the diabasic rocks than into the marble. The clean white marble is separated from the ore by only a few feet of garnet rock, while the diabase is garnetized, near the ore, and epidotized for tens of feet from the ore. The ore itself is so complete an alteration of whatever wall rock it replaces that there is no *prima facie* evidence of what it originally was. The pyritization is later than the principal period of ore deposition, the pyrite forming in fractures and cavities in the ore. The chalcopyrite accompanies the pyrite.

Surface action, producing partial hydration of the ore and considerable infiltration of secondary calcite, obscures the evidence.

Northeast of the mine workings is the largest exposure of marble found in the district. It is cut by sills of an entirely epidotized, basic, igneous rock. It is separated from the orebodies by massive garnet rock, or highly garnetized and silicified diabasic rock. The map (Fig. 22) shows the relation of orebodies to wall rock. The commercial ore is sharply bounded, but the mineralization is not.

The striking features in this mine are the absence of magnetite and of highly acid wall rocks, and the clear evidence that the diabasic rocks are more diffusely mineralized than the marble. At the same time the resemblance of the blocks of ore to engulfed limestone blocks or to roof pendants, is noticeable.

If the Ocania Mine were the only one studied, it would be difficult for the writer to defend his hypothesis. There are no aplitic or pegmatitic rocks in evidence. Except for one dike of doubtful age, no quartz-bearing igneous rock is exposed in the mine. There is abundant limestone.



OCANIA MINE
Fig. 22.

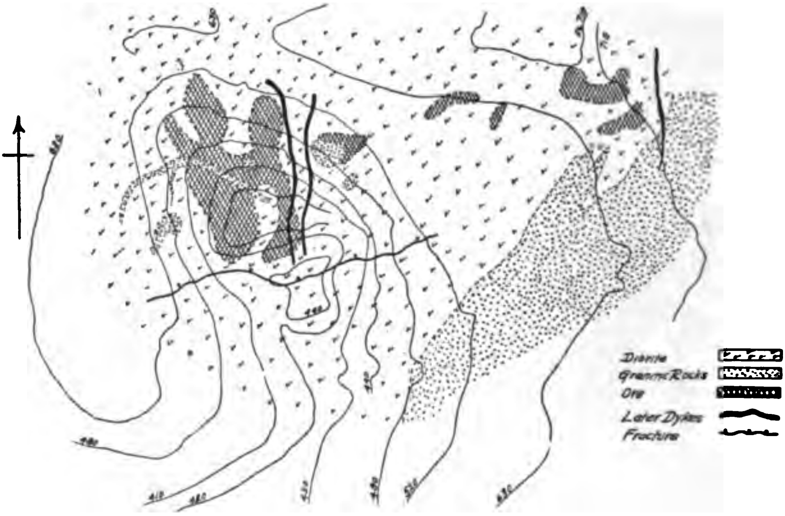
There are also some features that are not explained by any theory involving the diabase as the ore-bringing rock. The diffuse mineralization of the diabasic porphyry indicates a hydrothermal effect, later than the consolidation of the diabase. It is difficult to ascribe to the diabase porphyry a hematitizing effect on included limestone, or even a magnetitization free of chrome or titanium minerals, with a superimposed hematitization. The fact that the larger part of the marble has suffered no mineralization is also unexplained by any theory that makes the overwhelming of the limestone by the diabase the cause of mineralization.

On the other hand, if the solutions are supposed to come from the granite, the difficulties of explanation are not so great. A fissure in the diabase near the limestone, localized the ore and the mineralization in both formations. The distance of the ore exposed from the parent magma accounts for the preponderance of hematite over magnetite.

West Five Mine

This mine is the furthest west of the mines in the vicinity of Firmeza, and lies at an elevation of from 420 to 718 ft. There is a tunnel under the mine at a level of 299 ft.

The ore lies in flat or in steeply inclined lenses. It is almost entirely magnetite, the enclosing wall rock diorite, with considerable, more or less aplitic, granite through and near the ore. The diorite varies from the even, granular type shown in Fig. 8 to a porphyritic type. No fragmental diabasic rocks occur, nor is there any trace of limestone.



WEST 'S' MINE
FIG. 23.

The rocks from the aplitic dike, in the west side of the mine, were described under the granitic rocks. The connection between aplite and ore is shown on a small scale in the photograph (Fig. 17) of a small dike of aplite from the tunnel under the mine. The magnetite and epidote lying immediately next to the aplite, merge gradually into the granular diorite. There can be no question of any rocks being involved other than the diorite and the aplite. Their inter-relationship has been discussed in considering the question of genesis. Considerable garnet is found in the mine workings, and its occurrence is similar to the magnetite and epidote. In one small exposure the position of the garnet, as a contact effect in the diorite, next to the granite, is clearly shown. This is on the north side of the bottom level. The epidotization and silicification are more widely diffused from the granite than are the magnetite and garnet. One specimen, from the lowest level, showed an intergrowth of apatite with magnetite, and garnet and epidote, cut by later veinlets of epidote (Fig. 14). This is the only case in which apatite amounted to more than a very minor accessory.

On the south side of the mine is an east-west fault steeply inclined to the north. It is obviously post-mineral, and appears not to be of very great throw.

Summing up the evidence from this mine we find: no limestone; much aplitic or porphyritic granite; almost no specularite; but much magnetite, garnet, epidote, quartz—all in diorite.

The ore deposits of this mine offer the clearest evidence that limestone could not have been the dominant cause determining the locus of their deposition. There is no limestone in the immediate vicinity. The transition from ore to diorite is gradual. The mineralogy shows none of the well-crystallized calcite that is so abundant in the ore deposits that were obviously formed in limestone.

The intimate relation between ore and aplite is also well shown here. The dike exposed in the west side of the workings has led other observers to believe the granitic apophyses later than the ore, but as shown under the discussion of the various theories, the same phenomena are interpreted by the writer to indicate the contemporaneous formation of the ore and the aplite.

All the phenomena can be explained on the hypothesis advanced. That is that the aplite, the magnetite, and the lime silicates are all the effect of the one mineralizing period.

LOMA ALTA MINE

The Loma Alta workings lie at the top of a hill, just east of West Five Mine. They are at an elevation of from 900 to 1,000 ft. As they have been practically abandoned for some time, weathering agencies have obscured much of the evidence. Cropping out on the east side and pitching under the workings at a flat angle, is a sheet-like mass of porphyritic, pegmatitic granite. In the southwest corner of the workings is recognizable diabase porphyry. What is left of the ore is earthy hematite lying on the granite with some kaolin and chlorite. All around the workings on the hill are rounded boulders of more or less pure magnetite. The association of the granite, diabase and ore in this mine is suggestive, but the evidence is too obscured to be conclusive. The earthy hematite undoubtedly owes its origin to the weathering of the primary, crystalline iron oxides. There is no reason to believe that the process did not involve the formation of limonite and its dehydration.

WEST THREE MINE

An exposure in this mine, which lies on the same hill, south of and below Loma Alta Mine, is worth describing. It is a mass of magnetite, in diabase, directly next to a vertical pegmatitic granite dike. The rocks are all fairly unaltered, and the contact relationship well shown. There is no reason to suspect that any limestone was involved.

WEST FOUR MINE

This mine is located northeast of Loma Alta, at an elevation of from 645 to 935 ft. At the north end of the mine, in the upper levels, is a mass of clear white marble, entirely unmineralized. The wall rock in the upper levels is diabase porphyry; in the lower levels it is diorite. The diabase and diorite are both cut by somewhat porphyritic and pegmatitic granite. The ore is magnetite, and hematite and magnetite, and lies in lenses, whose major axes vary from a vertical to a horizontal position, on top of and next to the granite. These lenses occur mostly in the diorite, partly in the diabase, but not, so far as exposed, in contact with the marble. The entire series, diabase, diorite, granite, ore, is cut by a large dioritic dike. The mineralization effects are no different from those in West Five Mine. The chief difference is that there is marble exposed in the West Four workings. So far as can be determined it has had no direct effect on the formation of the ore deposits.

WEST ONE MINE

The West One workings are located north of Firmeza along the railroad tract. Exposed for a long time and not much worked of late years, they offer a poor field for study. Altered diabase and diorite, highly mineralized, are the wall rocks. Neither limestones nor granite are in evidence. From one of the cuts that is being worked, come the well-developed pyrite crystals, of which one is shown in Fig. 13. This exposure shows the pyrite all through the magnetite in cavities and along minute fractures. It establishes the age of the pyritization as definitely post-magnetitization.

Fig. 24 is a photograph of a drawing made with a camera lucida, from a thin section of a rock from West One Mine. The specimen is dark and dense, and shows patches of magnetite. In thin section the rock proves to be a diabasic porphyry, partly replaced by magnetite. The replacement is apparently independent of the mineralogy of the replaced rock. The magnetite occurs in patches in the groundmass, or in the phenocrysts, or partly in each.

West One Mine offers no new evidence in regard to the genetic problem. It does offer, however, an excellent field for the study of the pyritization.

EAST MINE

The East Mine itself lies east of Firmeza, at an elevation of from 440 to 700 ft., but it cannot be considered apart from the small cut just to the north at about 800 ft. elevation, known locally as the North Mine.

The large orebodies of this mine are made up of specularite and magnetite, with much epidote quartz and garnet. They lie in the lower levels of the mine around bosses of aplitic granite, in diorite and diabase porphyry. Fig. 18 is a photograph and Fig. 19 a microphotograph of a lime-iron garnet rim along a small aplite dike, in recognizable diabase porphyry. It shows the entire ability of the mineralizers that accompanied the aplite to garnetize the diabase.

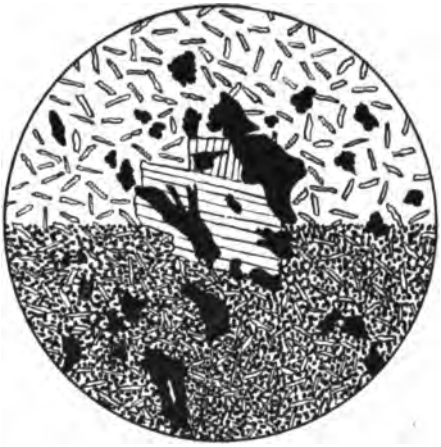


FIG. 24.—PHOTOGRAPH OF A DRAWING MADE WITH A CAMERA LUCIDA, FROM A THIN SECTION OF A SPECIMEN FROM WEST ONE MINE. IT SHOWS A DIABASE PORPHYRY, PARTLY REPLACED BY MAGNETITE (SOLID BLACK). THE PHENOCRYST IS PLAGIOCLASE. THE LATH-SHAPED CRYSTALS ARE PLAGIOCLASE. THE GROUNDMASS IS GRANULAR MAGNETITE AND CHLORITE.

(In the upper half of the drawing the interstitial magnetite has been left out.)

Fig. 26 is a microphotograph of a thin section of a completely epidoized and silicified diabase. The specimen comes from within 2 ft. to the one shown in Fig. 15. The two show successive stages in the hydrothermal metamorphism.

In the upper levels, extending toward the North Mine workings, are several included sheets of completely marmorized limestone. The North Mine workings, now abandoned, show remnants of orebodies. This ore is different from any found in the mines so far described. Well-developed rosettes of specularite blades in garnet quartz matrix; well-crystallized epidote in abundance; coarse quartz; much recrystallized calcite; in short all the components are found of a typical contact-metamorphic deposit in limestone.

In the East Mine, as in the others, later andesitic and dioritic dikes are in evidence. A northwest-southeast fault, pitching to the northeast, has cut the ore and aplitic granite as well as the older rocks. It has brought the most basic of the granitic rocks, the quartz-bearing diorite, in contact with the ore. That the specimen described as a wall rock of the East Mine by J. F. Kemp[39] came from this mass of quartz-bearing diorite seems probable. The rock in which the ore is found is of a much finer grain.

In the East Mine itself there is no evidence that limestone had any share in forming the orebodies. In the workings known as the North Mine, limestone is without doubt the host rock of the orebodies.

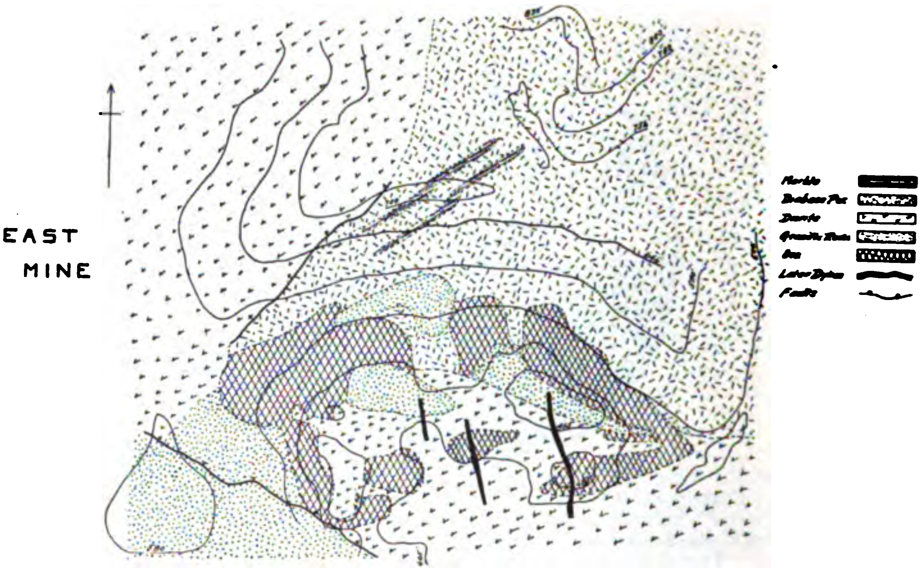


FIG. 25.

The ore in the main workings, with the associated granitic rocks, was formed at centers of intense mineralization. The ore deposits of the North Mine were formed at a greater distance from the centers of mineralization. The aplite dikes in the terrane between the two are the filling of the fissures through which the solutions traveled.

CHICHARRON (NORTH EAST) MINE

Located about ¾ mile northeast of the East Mine and now entirely idle, are the workings of the old Chicharron Mine. This mine is well north of the zone in which most of the mines lie, and in some respects differs from the others.

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[39] *Op. cit.*, p. 15.

The face of the mine is a cliff of diabase, over which the water of the Benevolencia River falls. This diabase is more or less mineralized, and replaced by magnetite, wollastonite, and epidote. The brook bed follows along the bottom of the cliff, and with an overgrown talus pile, separates the cliff from a mass of marble. Cutting the marble is a dike of magnetite, with interstitial wollastonite in small amounts (Fig. 27). A. C. Spencer describes this occurrence as follows:[40]

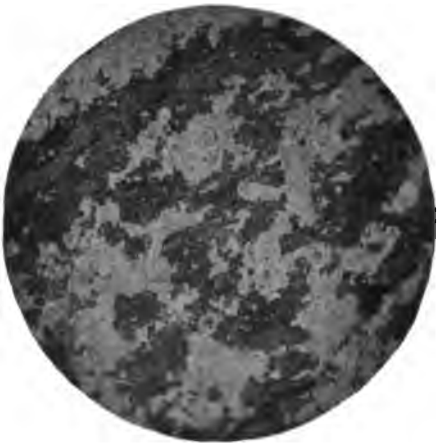

Fig. 26.—Diabase Porphyry Completely Replaced by Epidote and Quartz. From East Mine. × 18. Quartz—light. Epidote—dark.

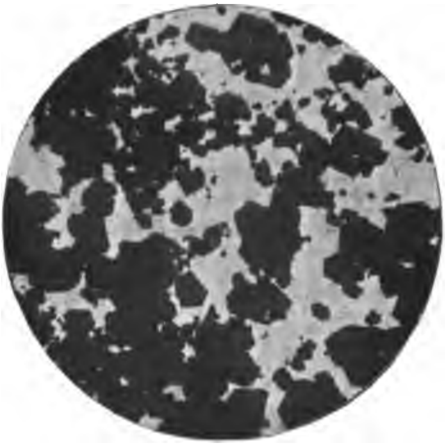

Fig. 27.—Ore from Chicharron Mine. × 18. Magnetite—black. Wollastonite—white.

> "At this place the magnetite seems to be an intrusive dike cutting across a mass of crystalline limestone into which it sends a short apophysis."

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[40] *Op. cit.*, p. 80.

The magnetite probably formed a dike, instead of a more diffuse form of deposit, because while the temperature was high, the pressure was not. As a result, the diffusive power of the mineralizers was not great, although they could form high-temperature minerals.

In the Chicharron Mine, also, the later dikes are in evidence.

Estancia Mines

To the east of East Mine, in the mineralized zone, lies Estancia Hill. All around the hill at from 700 to 800 ft. elevation lie mine workings. The ore in these workings is either magnetite and specularite replacing diabase, or it is the hematite-garnet-calcite combination that is eloquent of contact-metamorphic limestone, though no limestone is left. Around most of the hill, and all of the south side, is a railroad track at an elevation of about 500 to 550 ft., and it is nearly all cut in a porphyritic granite, in which micropegmatite predominates. In the valley north of Estancia Hill granite is exposed along the trail. The dump of an old tunnel run into Estancia Hill for 600 ft. at an elevation of 695 ft. is granite, and A. W. Gaumer, the Juragua Iron Co.'s chief engineer, informed the writer that all the tunnel was in the same rock. South of the railroad track runs the lower trail to Concordia, and it is all in diorite, or coarse diabase, which has been mapped as diorite.

All this evidence seems to indicate that a sheet of porphyritic granite intruded into diabasic rock with included marble, to form a sill. The orebodies at the upper contact of the granite formed in that rock to which the mineralizers had access.

Concordia Mine

This, the most easterly of the Juragua Iron Co.'s mines, offers no new evidence. It is the lowest of the mines, being at an elevation of from 300 to 450 ft. The rock and ore association is magnetite in diorite with granite near by.

The ore deposits are lenticular, and as a rule lie in an almost horizontal position. No limestone is in evidence.

VIII. GEOLOGIC HISTORY

The evidence from which the geologic history of the Firmeza district must be deduced is scanty. Almost all the field work was done in the immediate vicinity of the mines, and little search was made in the surrounding country for evidence which might give completeness to the geologic history. Enough material was found to make possible a sequential arrangement of the geologic events that produced the present topographic, petrologic, and mineralogic features of the district. These geologic events will be discussed in the order of their occurrence, as follows:

1. Sedimentation.
2. Igneous cycle (including ore deposition).
3. Uplift with tilting and erosion.
4. Deposition of coral limestone.
5. Emergence.
6. Submergence.

SEDIMENTATION

The formation which appears to be the oldest in the district is the marble. As has been shown in the discussion of the rocks, this marble is Mesozoic, and probably Cretaceous in age. The constituents of the floor upon which the limestones now represented by marble were deposited, cannot be determined; nor is it possible to estimate the thickness of these early sediments, because of the lack of any continuous section. The limestone found at the highest elevation is a tuff-limestone. Because of its position it may be considered as the youngest of these early sediments, and because of the volcanic fragments embedded in it at the time of its deposition, it may be considered as representing the connecting link between the period of sedimentation, and the initiation of the igneous cycle.

IGNEOUS CYCLE

The igneous cycle began with a period of volcanic activity, to which the clastic igneous rocks of basic nature bear witness. These volcanic rocks form the southern slopes of the Sierra Maestra range, and the tops of some of the foothills. They contain remnants of interbedded limestones, and must have formed contemporaneously with the latter part of the period of sedimentation.

The next event was the invasion of these volcanic rocks and sediments by a magma of basic composition. Of this "parent" magma the present representatives are the diabasic and dioritic rocks. These rocks were left in their present position partly by the chilling of the magma, and to a lesser degree, by intrusion into the sediments and volcanic rocks.

The invasion of basic magma was followed by a long period of differentiation. During this period the liquid residue in the magma chamber became more acidic, and charged with mineralizers. This acidic liquid, with its burden of mineralizers, invaded the earlier, chilled, basic, igneous rocks in the form of a granitic batholith.

Further differentiation, resulting in a concentration of the mineralizers, ended in the injection of the mineralizers, with admixed rock material, into cooling cracks in the basic rocks above the batholith.

There followed a separation of the highly heated mineralizers from their admixed rock material, which resulted in the formation of the ore deposits and the aplitic and pegmatitic apophyses.

It is uncertain to what part of the igneous cycle the marmorization of the limestone should be ascribed, because all the marble exposures are near or in igneous intrusive rocks, which range from diabase to granite. Any one of these may have been competent to produce the marmorization. It seems probable, however, that because it was the earliest, the invasion of the diabasic magma is responsible for the greater part of the marmorization.

What events were taking place at the surface during the igneous cycle is a question to which no answer has been found. The formation of cooling cracks, however, in the basic igneous rocks, seems to indicate that they were not very far from the surface at the time of ore deposition.

The last episode in the igneous cycle was the intrusion of basic dikes into fissures formed in all the igneous rocks and their associated ore deposits.

Uplift, Tilting, and Erosion

Because of the almost complete marmorization of the older limestones, their bedding is difficult to determine. But the pitch of the extrusive, and the few visible remnants of stratification in the marble, indicate a definite dip to the southeast. Evidence of uplift is found in the fact that limestone occurs at elevations up to 1,400 ft. above sea level.

As has been shown in the discussion of the areal geology, there is no conclusive evidence as to the exact time relation between the igneous cycle and the period of uplift and tilting, but because pressure is to a certain extent a function of depth, it seems probable that the ore deposits were all formed in an approximately horizontal plane. Their present position along an inclined plane would, in that case, be some measure of the differential elevation. Meager and inconclusive as the evidence is, it must be considered, and until evidence to the contrary is found, the uplift and tilting should be regarded as later than the igneous activity.

The coral limestones themselves show no evidence of tilting and this fact places the tilting as older than the coral limestones—that is pre-Pleistocene. Emergence has, however, continued.

Fragments of iron ore in the base of the later coastal limestones prove that this period of uplift was accompanied by erosion sufficient to cut into the ore deposits.

So far as the ore belt is concerned, the later record is one of continued erosion. Whatever else may have happened to that part of the district, no record remains except one of superficial alteration and erosion. The coral limestones and the valley topography show that dynamic forces were not entirely quiescent.

DEPOSITION OF CORAL LIMESTONE

The deposition of the Pleistocene-Recent coral limestone began when the land was about 350 ft. lower than at present. Since then there has been oscillation with predominant emergence.

EMERGENCE

The sea-cut cliffs in coral limestone, rising in terraces from the coast to an elevation of 350 ft., are evidence of the periodic emergence of the land since deposition of the limestone began. What the extent of the emergence was cannot be determined on the evidence in the district, because it has been masked by the most recent geologic event—submergence.

SUBMERGENCE

A. C. Spencer,[41] judging from other parts of the island, places the amount of the submergence at from 40 to 70 ft. If the thickness of the fluviatile deposits in the east-west valley could be determined, it would give an accurate measure of the amount of submergence in this district.

For reasons that have been discussed in detail under the interpretation of the topography, the writer believes that this submergence represents a movement of the sea.

The geologic history of this district is a record of quiet activity. There is no regional metamorphism or profound fracturing, such as would be expected if vast diastrophic movements had taken place. The only metamorphism that has left a record was due either to igneous or to superficial causes. The only fracturing recorded is the result of the shrinkage of igneous rocks in cooling, and gives evidence of little or no local movement. Igneous forces, acting through long periods of time, dominated the rock formations. Slow uplift, alternating with periods of almost complete quiet, and accompanied by much erosion, formed the topographic features.

IX. ECONOMIC APPLICATION

Any geological investigation of more than a purely academic interest must answer some questions of a commercial nature. The present study of the ore deposits of the Firmeza district, by its inquiry into their origin and extent, is able to answer certain questions which are of economic interest. These questions refer to the continuation of the deposits in depth, a possible change in their composition if they do extend downward, and the favorable locus for the search for more ore.

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[41] *Op. cit.*, p. 34.

CONTINUATION OF THE OREBODIES IN DEPTH

Since the mineralization came from below, from the granitic massif, there is no reason to believe that the ore will not extend at least to the granite. On the other hand, there is no reason to believe that the separation of the mineralizers from the granite was incomplete, so that no ore can be looked for in the granite. Where the ore lies on top of sheet-like granitic masses, like the Estancia and Loma Alta deposits, a definite limit in depth is in sight, but in workings where the massive granite is not in evidence, extensive development in depth seems probable.

CHARACTER OF THE ORE IN DEPTH

The pyritization in the ores at Firmeza has made the owners of the ore deposits fear that the ores would change to pyrite in depth. However, the pyritization is later and independent of depth. There is no reason to believe that the ores found at greater depth will be any higher in sulphur than those now being mined in the lower levels of the larger mines. The lower sulphur content of the ores close to the surface is due to superficial oxidation of the pyrite. Below the level of surface oxidation the sulphur content may be expected to remain fairly constant.

The only change that seems probable is an increase in the proportion of magnetite to hematite, as the workings approach the deeper centers of mineralization.

FAVORABLE LOCUS FOR EXPLORATION WORK

The problem of exploration work may be considered in two ways—positively and negatively. Both are of fundamental importance, although frequently the question of where to look for more ore is considered, while the equally important question of where not to look for ore is neglected. It is just as necessary to discover in which localities the search for ore will be fruitless and therefore any exploration work would be valueless, as it is to know where the search for ore will be profitable.

Obviously, rocks formed later than the ore, like the coastal limestone, cannot be expected to contain ore deposits. The granitic massif, from which the ore solutions are thought to have escaped, is also to be regarded as barren ground.

On the other hand, the granitic apophyses, aplites and pegmatitic granite are the best guides as to the location of orebodies. It is their upper surface which merits the most careful search.

Since the orebodies contain much magnetite, the magnetic survey is undoubtedly the best guide to their location. But a careful study of the formations is necessary to interpret such a survey, and to decide what areas of magnetic attraction are most valuable, and also what areas, in spite of a lack of this attraction, offer fields for development.

The economic application of geologic evidence will be successful only as the interpretation of the geologic phenomena is correct. At the same time the ultimate test of any geologic theory is its application in the field.

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TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Effect of Sulphur on Low-Carbon Steel

BY CARLE R. HAYWARD,* S. B., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

SULPHUR has long been one of the banes of the steel manufacturer and often no effort and expense have been spared in order to reduce it to a small per cent. in the finished product. This condition is due to a general conviction that in many cases where steels have failed in service, sulphur has been the cause. But there has been a growing feeling in recent years that the verdict against sulphur has been unnecessarily severe. In cases of segregation it was present in augmented amounts along with other impurities, but it had not caused the segregation. High sulphur in pig iron is caused by poor furnace conditions and the sulphur is merely one indication of an iron that has not been properly reduced. No amount of subsequent treatment under oxidizing conditions in the open-hearth furnace can remedy the defects, although the per cent. of sulphur may be considerably reduced. In other words, the causes of bad steel can frequently be traced back to bad pig iron, and sulphur is merely one indication that the pig iron is bad. The writer recently visited a steel plant where a mass of evidence had been accumulated which substantiated this fact, and the superintendent was emphatic in stating that high sulphur was not harmful provided the steel was not otherwise poor due to insufficient reduction in the blast furnace.

The presence of a moderate amount of sulphur is desirable from the standpoint of the man who machines the steel. The low-sulphur material drags and the production of a smooth surface is very difficult. A slight increase in sulphur enables the machinist to produce a smooth surface without difficulty.

Since, therefore, such large quantities of steel are subjected to machining, it becomes highly important that the sulphur controversy should be settled, and if its presence is proved to be harmless the ban on it should be lifted.

Among the recent papers on the effect of sulphur on steel is one by Dr. J. S. Unger, Manager of the Central Research Bureau, Carnegie Steel

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Co.[1] The results of an exhaustive series of tests are given and the conclusion states: "The author does not advocate paying no attention whatever to sulphur content in steel but believes firmly that a steel containing less than 0.100 per cent. is not necessarily bad, and that it will show little, if any, difference in quality when compared with the same steel of much lower sulphur, other conditions being the same."

The present investigation was undertaken at the suggestion of A. H. Annan of the Rhode Island Tool Co., who coöperated by furnishing the steel and machining the specimens. The results are presented as a contribution to the general knowledge on the subject.

Steels Used

It was planned to use three steels of different sulphur content but with the other elements the same. The manganese was an exception, however, for with this element part is in the form of MnS existing free in the steel and the remainder is dissolved in the steel. It is evident, therefore, that the manganese should vary but that the amount in excess of MnS should be constant in the different steels.

The steels finally selected were in the form of ¾-in. round bars. Two bars of each grade were required to furnish sufficient specimens. The analyses are shown in Table 1.

TABLE 1

Mark	Carbon, Per Cent.	Total Manganese, Per Cent.	Excess Manganese, Per Cent.	Phosphorus, Per Cent.	Silicon, Per Cent.	Sulphur, Per Cent.
1	0.18	0.55	0.48	0.007	0.01	0.038
1A	0.18	0.57	0.50	0.009	0.02	0.041
2	0.17	0.67	0.52	0.008	0.01	0.086
2A	0.18	0.70	0.55	0.010	0.03	0.087
3	0.18	0.80	0.54	0.006	0.02	0.152
3A	0.17	0.80	0.55	0.011	0.03	0.148

Heat Treatment

In order to make a comparison of the steels under different conditions it was decided to heat all the specimens to a temperature just above the critical range, quench in water, and reheat different lots to 300°, 400°, 500° and 600°C. respectively. For this purpose the bars were cut into 7-in. lengths, which was sufficient for tensile specimens and specimens for microscopic examination.

The furnace used is shown in Fig. 1. The muffle is made of alundum and is 2 in. high, 9 in. wide and 16 in. long. It is wound with No. 15 excello resistance wire.

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[1] *Iron Age*, No. 97, pp. 146–150 (1916).

In order to obtain uniform heating of the specimens, they were supported on an asbestos rack, as shown. Asbestos shields were placed on the sides and ends of the rack so that the specimens were practically in a muffle within a muffle.

A platinum platinum-rhodium thermocouple was introduced through the back of the muffle into the center of the heating chamber and connected to a Siemens and Halske recording galvanometer.

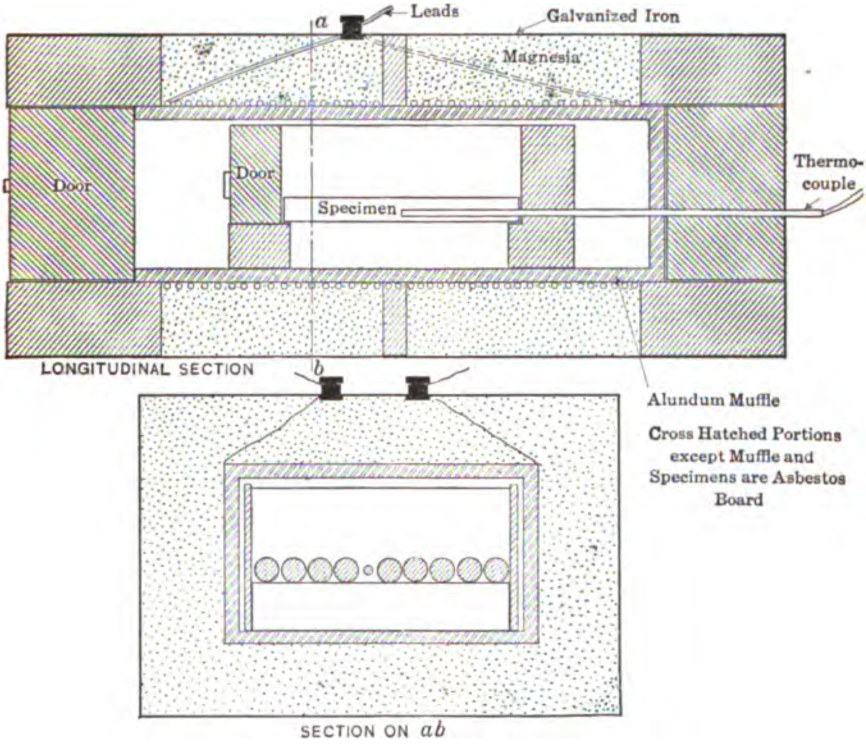
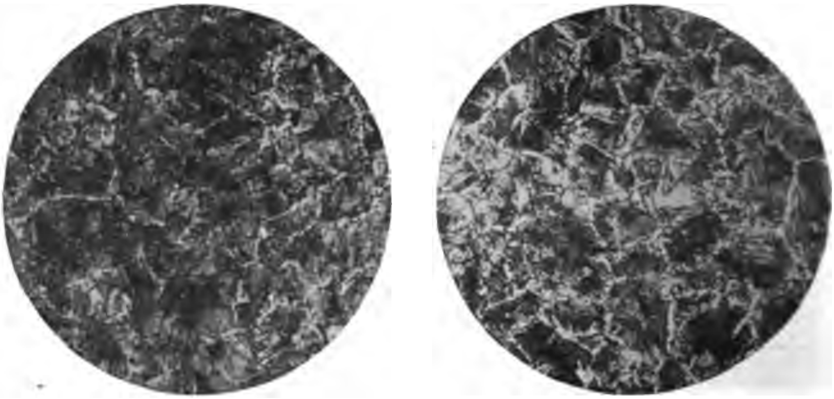


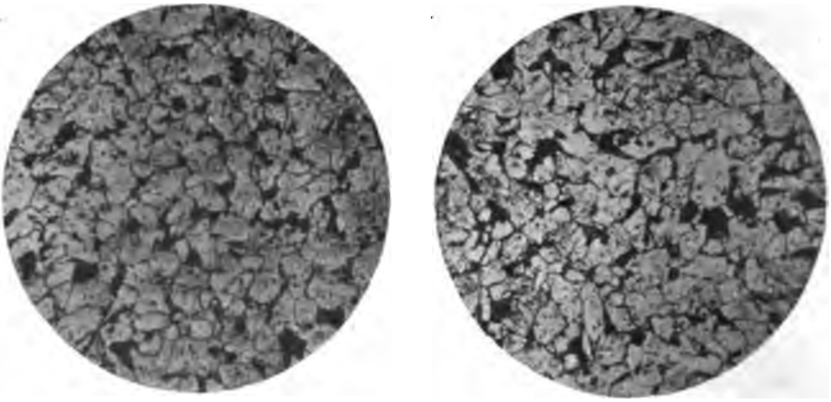
Fig. 1.—Heat-Treatment Furnace.

The procedure in heat treatment was as follows: The furnace was heated to 880° C. and nine specimens introduced. This caused the temperature to fall to 550° C. and it took about 40 min. to again reach 880°. When the latter temperature was reached it was maintained constant for 15 min., after which the specimens, except as noted below, were quenched in water and a new lot introduced into the furnace. This was continued until all but 18 of the specimens had been treated. Nine of these, consisting of three high-, three medium- and three low-sulphur steels, were removed from the furnace and allowed to cool in air. The



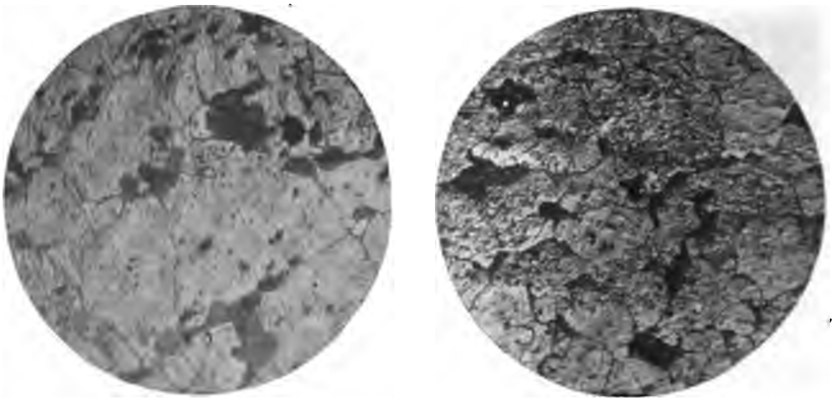
a.—**High-Sulphur Steel.** *b.*—**Low-Sulphur Steel.**

Quenched in Ice Water.



c.—**High-Sulphur Steel.** *d.*—**Low-Sulphur Steel.**

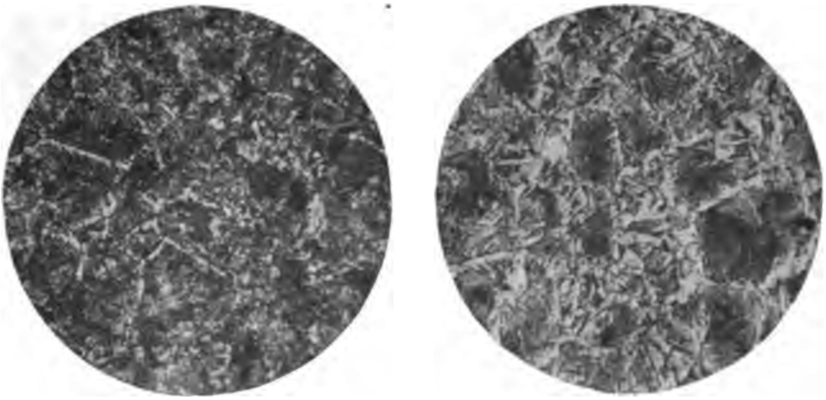
Cooled in Air.



e.—**High-Sulphur Steel.** *f.*—**Low-Sulphur Steel.**

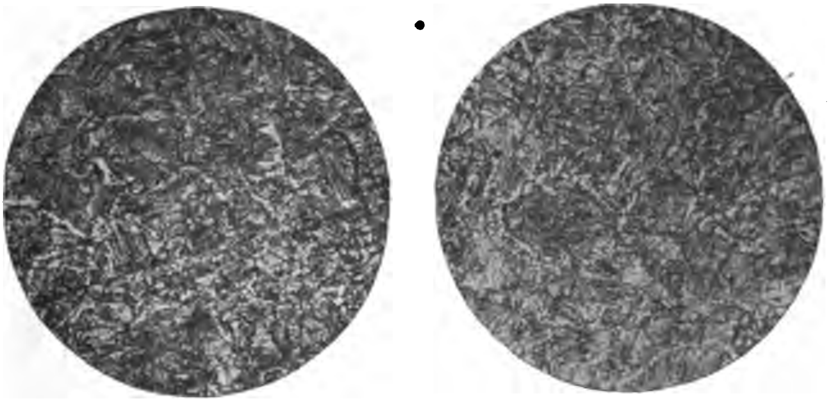
Cooled in Furnace.

PLATE I.



a.—**High-Sulphur Steel.** *b.*—**Low-Sulphur Steel.**

Reheated to 300°.



c.—**High-Sulphur Steel.** *d.*—**Low-Sulphur Steel.**

Reheated to 500°.

Plate II.

Table 2

No.	Mark		Yield Point, Lb. per Sq. In.	Breaking Load, Lb. per Sq. In.	Elongation, Per Cent. in 2 In.	Reduction of Area, Per Cent.
1	1A	Heated to 880° C. and quenched in ice water.	58,500	86,250	23.0	64.7
2	1A	Heated to 880° C. and quenched in ice water.	58,000	85,250	23.5	62.4
3	1A	Heated to 880° C. and quenched in ice water.	57,250	85,500	23.5	62.4
Average			57,900	85,600	23.3	63.1
4	2A	Heated to 880° C. and quenched in ice water.	53,000	79,750	26.0	69.3
5	2A	Heated to 880° C. and quenched in ice water.	56,250	77,000	29.5	71.4
6	2A	Heated to 880° C. and quenched in ice water.	51,500	76,250	27.5	69.3
Average			53,600	77,700	27.7	70.0
7	3A	Heated to 880° C. and quenched in ice water.	56,500	84,250	22.0	59.9
8	3A	Heated to 880° C. and quenched in ice water.	79,250*	89,250	22.5	59.9
9	3A	Heated to 880° C. and quenched in ice water.	57,300	84,500	22.0	62.4
Average			56,900	86,000	22.2	60.7
10	1A	Heated to 880° and cooled in still air	41,000	58,500	41.0	67.0
11	1A	Heated to 880° and cooled in still air	46,750	58,500	41.5	69.3
12	1A	Heated to 880° and cooled in still air	41,750	58,250	41.5	67.0
Average			43,150	58,400	41.3	67.8
13	2A	Heated to 880° and cooled in still air	45,750	58,000	41.0	67.0
14	2A	Heated to 880° and cooled in still air	46,250	58,000	40.5	69.3
15	2A	Heated to 880° and cooled in still air	39,750*	58,000	41.5	69.3
Average			46,000	58,000	41.3	68.5
16	3A	Heated to 880° and cooled in still air	40,000	59,500	38.0	62.4
17	3A	Heated to 880° and cooled in still air	41,500	59,500	39.5	64.7
18	3A	Heated to 880° and cooled in still air	46,250	59,250	39.5	64.7
Average			42,600	59,400	39.0	63.9
19	1A	Heated to 880° and cooled in furnace	38,000	52,500	40.5	62.4
20	1A	Heated to 880° and cooled in furnace	37,750	52,750	40.5	64.7
21	1A	Heated to 880° and cooled in furnace	33,750	52,500	40.0	62.4
Average			36,500	52,600	40.3	63.2
22	2A	Heated to 880° and cooled in furnace	37,000	53,000	40.0	62.4
23	2A	Heated to 880° and cooled in furnace	38,750	53,000	40.0	62.4
24	2A	Heated to 880° and cooled in furnace	34,500	53,000	41.0	59.9
Average			36,750	53,000	40.3	61.6
25	3A	Heated to 880° and cooled in furnace	37,500	55,000	37.5	59.9
26	3A	Heated to 880° and cooled in furnace	35,500	55,000	38.5	59.9
27	3A	Heated to 880° and cooled in furnace	32,250	55,000	38.0	69.3*
Average			35,080	55,000	38.0	63.0
301	1	Quenched at 880°. Reheated to 300°	53,750	76,000	25.5	69.3
302	1	Quenched at 880°. Reheated to 300°	58,000	76,000	26.0	69.3
303	1	Quenched at 880°. Reheated to 300°	56,000	75,000	28.0	71.4
Average			55,900	75,700	26.5	70.0
304	2	Quenched at 880°. Reheated to 300°	57,500	73,250	28.5	71.4
305	2	Quenched at 880°. Reheated to 300°	61,250	78,000	26.5	71.4
306	2	Quenched at 880°. Reheated to 300°	59,750	73,750	25.0	71.4
Average			59,500	75,000	26.7	71.4
307	3	Quenched at 880°. Reheated to 300°	57,000	80,250	23.5	62.4
308	3	Quenched at 880°. Reheated to 300°	56,500	83,000	21.5	59.9
309	3	Quenched at 880°. Reheated to 300°	59,000	85,000	19.5	59.9
Average			57,500	82,750	21.5	60.7
401	1	Quenched at 880°. Reheated to 400°	63,750	76,500	28.5	71.4
402	1	Quenched at 880°. Reheated to 400°	58,500	76,000	29.5	73.5
403	1	Quenched at 880°. Reheated to 400°	58,750	76,450	28.5	71.4
Average			60,300	76,300	28.8	72.1
404	2	Quenched at 880°. Reheated to 400°	62,750	75,000	26.5	69.3
405	2	Quenched at 880°. Reheated to 400°	63,750	80,000	24.0	69.3
406	2	Quenched at 880°. Reheated to 400°	61,250	74,750	28.5	71.4
Average			62,600	76,600	26.3	70.0
407	3	Quenched at 880°. Reheated to 400°	64,000	84,000	21.5	62.4
408	3	Quenched at 880°. Reheated to 400°	62,000	82,750	21.5	62.4
409	3	Quenched at 880°. Reheated to 400°	61,250	82,750	24.0	62.4
Average			62,400	83,150	22.3	62.4
501	1	Quenched at 880°. Reheated to 500°	55,250	70,750	33.5	73.5
502	1	Quenched at 880°. Reheated to 500°	58,250	73,250	31.5	73.5
503	1	Quenched at 880°. Reheated to 500°	57,500	74,000	31.0	73.5
Average			57,000	72,650	32.0	73.5
504	2	Quenched at 880°. Reheated to 500°	60,750	73,250	29.0	71.4
505	2	Quenched at 880°. Reheated to 500°	61,250	74,000	32.0	71.4
506	2	Quenched at 880°. Reheated to 500°	57,000	72,500	26.0	71.4
Average			59,650	73,250	29.0	71.4
507	3	Quenched at 880°. Reheated to 500°	61,500	78,750	26.0	67.0
508	3	Quenched at 880°. Reheated to 500°	60,500	77,500	26.0	67.0
509	3	Quenched at 880°. Reheated to 500°	60,750	77,750	26.0	67.0
Average			60,900	78,000	26.0	67.0
601	1	Quenched at 880°. Reheated to 600°	53,500	69,250	35.0	75.5
602	1	Quenched at 880°. Reheated to 600°	51,000	67,750	35.0	75.5
603	1	Quenched at 880°. Reheated to 600°	52,000	69,500	32.5	75.5
Average			52,150	68,850	34.2	75.5
604	2A	Quenched at 880°. Reheated to 600°	54,500	70,500	35.5	73.5
605	2A	Quenched at 880°. Reheated to 600°	52,500	72,000	32.5	73.5
606	2A	Quenched at 880°. Reheated to 600°	53,750	70,750	31.0	73.5
Average			53,550	71,100	33.0	73.5
607	3	Quenched at 880°. Reheated to 600°	55,000	74,250	31.0	71.4
608	3	Quenched at 880°. Reheated to 600°	55,000	73,750	30.5	71.4
609	3	Quenched at 880°. Reheated to 600°	55,500	75,750	30.0	69.3
Average			55,150	74,600	30.5	70.7

* Results abnormal. Not included in average.

remaining nine, consisting of three of each sulphur content, were allowed to cool in the furnace.

In drawing the quenched specimens at the various temperatures, the procedure was as follows: The furnace was heated to 600° C. and nine specimens (three of each sulphur content) were introduced. The temperature fell and it required about 30 min. to come back to 600°. It was maintained here for 10 min. and then the specimens were withdrawn and quenched in water. The furnace was cooled to 500° C. and another set of nine specimens was introduced. When the temperature had regained 500° it was maintained constant for 10 min. and then the specimens were withdrawn and quenched. Following a similar procedure, sets of nine specimens were treated at 400° and 300° C. respectively.

Tensile Tests

After sawing off ¾ in. from each piece for microscopic examination, standard test specimens were prepared with 2-in. gage length, 0.505-in. diameter and threaded ends. These were pulled in an Olsen machine in the testing laboratory of the Massachusetts Institute of Technology.

The results obtained are given in Table 2.

For more ready comparison, the averages are retabulated in Table 3. The designations *L*, *M* and *H* refer to low, medium and high sulphur content.

TABLE 3

		Yield Point, Lb. per Sq. In.	Breaking Load, Lb. per Sq. In.	Elongation, Per Cent. in 2 In.	Reduction of Area, Per Cent.
L	Heated to 860° C. and quenched in ice water	57,900	85,600	23.5	63.1
M	Heated to 860° C. and quenched in ice water	53,600	77,700	27.7	70.0
H	Heated to 860° C. and quenched in ice water	56,900	86,000	22.0	60.7
L	Heated to 860° C. and cooled in still air	43,200	58,400	41.5	67.8
M	Heated to 860° C. and cooled in still air	46,000	58,000	41.3	68.5
H	Heated to 860° C. and cooled in still air	42,600	59,400	39.0	63.9
L	Heated to 860° C. and cooled in furnace	36,500	52,600	40.5	63.2
M	Heated to 860° C. and cooled in furnace	36,800	53,000	40.3	61.6
H	Heated to 860° C. and cooled in furnace	35,100	55,000	38.0	63.0
L	Quenched at 860° in ice water. Reheated to 300°	55,900	75,700	26.5	70.0
M	Quenched at 860° in ice water. Reheated to 300°	59,500	75,000	26.7	71.4
H	Quenched at 860° in ice water. Reheated to 300°	57,500	82,750	21.5	60.7
L	Quenched at 860° in ice water. Reheated to 400°	60,300	76,300	28.8	72.1
M	Quenched at 860° in ice water. Reheated to 400°	62,600	76,600	26.3	70.0
H	Quenched at 860° in ice water. Reheated to 400°	62,400	83,200	22.3	62.4
L	Quenched at 860° in ice water. Reheated to 500°	57,000	72,700	32.0	73.5
M	Quenched at 860° in ice water. Reheated to 500°	59,700	73,300	29.0	71.8
H	Quenched at 860° in ice water. Reheated to 500°	60,900	78,000	26.0	67.0
L	Quenched at 860° in ice water. Reheated to 600°	52,200	68,800	34.2	75.5
M	Quenched at 860° in ice water. Reheated to 600°	53,600	71,100	33.0	73.5
H	Quenched at 860° in ice water. Reheated to 600°	55,200	74,600	30.5	70.7

There is not sufficient variation in the results to make an effective plot, but Table 4 summarizes the figures by giving the order in which they occur from high to low.

TABLE 4

	Yield Point			Breaking Load			Elongation			Reduction of Area		
	L	*M*	*H*	*L*	*M*	*H*	*L*	*M*	*H*	*L*	*M*	*H*
Quenched in ice water..	1	3	2	2	3	1	3	1	2	3	1	2
Cooled in still air......	2	1	3	2	3	1	1	2	3	2	1	3
Cooled in furnace	2	1	3	3	2	1	1	2	3	2	3	1
Reheated to 300°......	3	1	2	2	3	1	3	1	2	3	1	2
Reheated to 400°......	3	1	2	3	2	1	1	2	3	1	2	3
Reheated to 500°......	3	2	1	3	2	1	1	2	3	1	2	3
Reheated to 600°......	3	2	1	3	2	1	1	2	3	1	2	3
Totals..............	17	11	14	18	17	7	11	12	19	13	12	17
Order of totals......	3	1	2	3	2	1	1	2	3	2	1	3

Photomicrographs

One specimen from each set was polished, etched in a solution of 10 per cent. HNO_3 in alcohol and examined under the microscope. The photographs show a representative spot on each specimen magnified 300 diameters.

The specimens were studied at higher magnifications than those given in the micrographs, but no further information was obtained. The lower magnifications gave a better idea of the general structure and, therefore, only these were reproduced in the paper.

Shock Tests

After concluding the tensile tests, it was thought desirable to determine the effect of varying sulphur content on specimens subjected to shock.

Through the kindness of the testing department of the Watertown Arsenal, the use of their Charpy machine was obtained for this purpose.

The remaining ¾-in. stock of bars marked 1, 2 and 3 was sawed into 2-in. lengths and the specimens subjected to heat treatment in sets of three under the same conditions that obtained in the case of the tensile specimens. After heat treatment, the specimens were machined to conform to the following specifications:

Length 55 mm., cross-section 10 by 10 mm., notch across one side, midway between the ends, 1 mm. wide. Radius of cutter edge, ⅔ mm.

The Charpy machine consists of a heavy pendulum which drops from a fixed height, strikes the specimen supported at each end and breaks

TABLE 5

Treatment	No.	Mark	Breaking Shock, Ft.-Lb. per Sq. In.	Average
Heated to 480° and quenched in cold water	1	1A	494	526
Heated to 480° and quenched in cold water	2	1A	545	
Heated to 480° and quenched in cold water	3	1A	540	
Heated to 480° and quenched in cold water	4	2A	523	518
Heated to 480° and quenched in cold water	5	2A	488	
Heated to 480° and quenched in cold water	6	2A	542	
Heated to 480° and quenched in cold water	7	3A	432	470
Heated to 480° and quenched in cold water	8	3A	507	
Heated to 480° and quenched in cold water	9	3A	470	
Heated to 480° and cooled in still air	10	1A	356*	456
Heated to 480° and cooled in still air	11	1A	460	
Heated to 480° and cooled in still air	12	1A	452	
Heated to 480° and cooled in still air	13	2A	512	519
Heated to 480° and cooled in still air	14	2A	521	
Heated to 480° and cooled in still air	15	2A	524	
Heated to 480° and cooled in still air	16	3A	418	416
Heated to 480° and cooled in still air	17	3A	398	
Heated to 480° and cooled in still air	18	3A	433	
Heated to 480° and cooled in furnace	19	1A	258	279
Heated to 480° and cooled in furnace	20	1A	284	
Heated to 480° and cooled in furnace	21	1A	295	
Heated to 480° and cooled in furnace	22	2A	353	355
Heated to 480° and cooled in furnace	23	2A	357	
Heated to 480° and cooled in furnace	24	2A	259*	
Heated to 480° and cooled in furnace	25	3A	283	273
Heated to 480° and cooled in furnace	26	3A	256	
Heated to 480° and cooled in furnace	27	3A	279	
Quenched at 480°. Reheated to 300°	301	1A	633	605
Quenched at 480°. Reheated to 300°	302	1A	447*	
Quenched at 480°. Reheated to 300°	303	1A	578	
Quenched at 480°. Reheated to 300°	304	2A	562	543
Quenched at 480°. Reheated to 300°	305	2A	537	
Quenched at 480°. Reheated to 300°	306	2A	530	
Quenched at 480°. Reheated to 300°	307	3A	462	439
Quenched at 480°. Reheated to 300°	308	3A	422	
Quenched at 480°. Reheated to 300°	309	3A	434	
Quenched at 480°. Reheated to 400°	401	1A	613	597
Quenched at 480°. Reheated to 400°	402	1A	597	
Quenched at 480°. Reheated to 400°	403	1A	582	
Quenched at 480°. Reheated to 400°	404	2A	540	546
Quenched at 480°. Reheated to 400°	405	2A	541	
Quenched at 480°. Reheated to 400°	406	2A	556	
Quenched at 480°. Reheated to 400°	407	3A	474	450
Quenched at 480°. Reheated to 400°	408	3A	447	
Quenched at 480°. Reheated to 400°	409	3A	429	
Quenched at 480°. Reheated to 500°	501	1A	439*	682
Quenched at 480°. Reheated to 500°	502	1A	694	
Quenched at 480°. Reheated to 500°	503	1A	670	
Quenched at 480°. Reheated to 500°	504	2A	560	561
Quenched at 480°. Reheated to 500°	505	2A	572	
Quenched at 480°. Reheated to 500°	506	2A	552	
Quenched at 480°. Reheated to 500°	507	3A	495	473
Quenched at 480°. Reheated to 500°	508	3A	456	
Quenched at 480°. Reheated to 500°	509	3A	469	
Quenched at 480°. Reheated to 600°	601	1A	726	727
Quenched at 480°. Reheated to 600°	602	1A	728	
Quenched at 480°. Reheated to 600°	603	1A	726	
Quenched at 480°. Reheated to 600°	604	2A	584	597
Quenched at 480°. Reheated to 600°	605	2A	604	
Quenched at 480°. Reheated to 600°	606	2A	602	
Quenched at 480°. Reheated to 600°	494	3A	494	506
Quenched at 480°. Reheated to 600°	502	3A	502	
Quenched at 480°. Reheated to 600°	521	3A	521	

* Abnormal. Not included in average.

it at the notch. The pendulum then continues its swing and the height it reaches is registered. Knowing the weight of the pendulum, the height it falls and the height it rises, a simple calculation gives the energy consumed in breaking the specimen.

The results obtained are given in Table 5.

Conclusions

Table 4, which expresses the summary of the tensile tests, shows that the high-sulphur steel has for each treatment the highest breaking load while the yield point ranks first for two treatments, second for three and third for two. From this we may conclude that the sulphur does not lower the tensile strength.

The figures for elongation and reduction of area show that there is little difference in ductility between the low- and medium-sulphur steels, but the ductility of the high-sulphur steel is slightly lower than the other two for most of the treatments.

The average figures for the shock tests, except for the air- and furnace-cooled specimens, are highest for each treatment in the case of the low-sulphur steels and lowest for each treatment for the high-sulphur steels. The widest difference appears in the steels which have been quenched and reheated.

It is difficult to draw definite conclusions from the results because of the newness of the shock test and the difference of opinion among engineers regarding its value. The tensile tests are not unfavorable to steels with moderate amounts of sulphur, while the shock tests show a decided falling off in strength as the sulphur increases. Until the interpretation of the results from the Charpy machine is more fully understood, it is impossible to say to which set of tests the most importance should be attached.

Further light might be thrown on the subject by making alternate stress or fatigue tests. It would be important to learn whether the results would confirm the tensile or shock tests. Unfortunately, however, the stock of steels used in the previous work was exhausted, and whatever the results of the fatigue tests there would be an uncertainty in their interpretation because of difference in stock. It was, therefore, decided not to include this series in the present investigation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close April 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Recrystallization after Plastic Deformation

BY HENRY M. HOWE, LL. D.,* BEDFORD HILLS, N. Y.

(New York Meeting, February, 1917)

THIS paper is a discussion of the extremely valuable one of Mathewson and Phillips, The Recrystallization of Cold-Worked Alpha Brass on Annealing,[1] which not only gives us a wealth of important data reached with great intelligence, but also shows both uncommon powers of imagination and a perfectly fair spirit.

Let me try to analyze some of their results, and to enunciate some of the laws to which these results point. For brevity I refer to them as "The Authors." The page numbers in parentheses refer to their paper.

1. *The Visible Aspects of Recrystallization.*—Plastic deformation in a single direction, as in cold-rolling, draws the grains out in that direction, that is, inequiaxes them, without destroying their apparent individuality. Each grain seems to endure the drawing out without changing either its volume or its orientation materially, for the etching tint of each grain seems to be as uniform after as before the drawing out, save for the "etch bands,"[2] shown in Figs. *A* to *D* of the Authors' Plate I.

As the drawn-out metal is heated progressively, on reaching a certain "disintegration temperature" these old grains seem to break up, probably into submicroscopic fragments, and this apparent disintegration is followed by a coalescence of these fragments, so that their number decreases and their size increases, till they may become very much coarser than before the drawing out. Ruder[3] coarsened the grains of silicon steel in this way till single grains reached an area of 50 cm. (7.75 sq. in.).

The deformation itself probably breaks each grain up into many very small fragments, but the fact that each grain continues to have a uniform etching tint different from that of its neighbors indicates that these fragments retain the initial common orientation of the grain. The apparent disintegration which we observe, then, is rather an abandonment of community of orientation for individuality of orientation, by the fragments formed earlier during the deformation. Hence we should speak rather of a "reorientation" than of a "disintegration" temperature. The reorientation and regrowth may jointly be called "recrystallization."

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[1] *Bulletin* No. 109, p. 1 (January, 1916).

[2] Rosenhain calls these "slip bands," and I have called them "X bands," to avoid any preconception as to their nature (*The Metallography of Steel and Cast Iron*, p. 452).

[3] *Trans.*, vol. 47 p. 569.

The recrystallization is accompanied by softening, that is to say, though the first raising of the temperature after the cold-rolling may increase the hardening which the cold-rolling caused, yet with further rise of temperature the hardness falls progressively to that of the normal metal.

While the Authors would explain the nature of disintegration and regrowth, their main objective seems to be to trace the connection between these two processes and this softening.

2. *Our natural conception of disintegration and grain growth* I take to be that the plastic deformation itself breaks each grain up into many fragments as already pointed out, of which the number increases and the size decreases as the deformation increases; that each fragment retains for the time the initial orientation of the grain from which it is broken, as is shown by the uniform etching tint of all these fragments of a given grain; but that each is sheathed completely with metal made amorphous by the friction during the act of deformation. This amorphous metal cements the whole together. On this hypothesis when the deformed metal is heated, five distinct changes occur side by side, though not necessarily simultaneously or in any fixed proportion to each other. These are, first, a progressive reabsorption of the amorphous sheaths by the crystalline fragments which they inclose; second, a suspected progressive but undescribed change in the condition of so much of these sheaths as has not yet become reabsorbed, or of the crystalline grain fragments thus sheathed, or of both; third, an apparent disintegration of the initial grains, which is really only a reorientation of the grain fragments already formed; fourth, their progressive growth by coalescence, with consequent lessening of their number; and fifth, the progressive release of stress as the mobility increases and the existing elastic limit falls correspondingly. The first three may be grouped together as "recrystallization." Let us in (3) to (7) glance at the evidence that these five agencies are at work.

3. *The reabsorption of the sheaths* with rising temperature is inferred from the observation that, by heating before etching, the etch bands, which seems to represent an accumulation of this same amorphous metal along the slipping planes, are made first fainter and finally undetectable.

4. *The reorientation of the grain fragments* is shown by the apparent disintegration which the grains undergo, for this disintegration means only that the several fragments of a given grain cease to have common orientation and hence to etch alike, and instead adopt each a new orientation, so that they etch differently and hence appear as distinct grains or grain fragments. It is not clear that this distinction between grains

and grain fragments need be insisted upon in what follows, for indeed each fragment seems for all intents and purposes to be an independent grain, once its reorientation has occurred.

5. *Grain Growth.*—The progressive growth of the grain fragments is familiar to all through the coarsening that occurs on heating cold-rolled or otherwise plastically deformed metal.

6. *The progressive release of stress* could no doubt be proved by noting how much, after reheating to various temperatures, the specimen bends out of shape spontaneously on cutting away a fixed part of it, so as to destroy the balance of internal stresses remaining after the initial cold-rolling itself. We may assume confidently that the degree of this spontaneous bending decreases progressively as the temperature of annealing, prior to cutting away, rises.

7. *A progressive change in the condition of the amorphous sheaths* or of the crystalline fragments is neither shown nor known. I have inferred it as a probable cause of the perturbations which the cohesion, as indicated by the elastic limit and hardness of cold-rolled iron, both pure and cementite-bearing (mild steel), undergoes on gentle heating. On either slight heating or mere rest such iron may soften materially or may harden greatly, according to the intensity of the previous deformation.[4]

8. *In what Proportion is the Softening Caused by Annealing due to Each of These Five Influences?*—Though the present evidence does not enable us to give a full answer to this question, some inferences concerning it may be drawn. Let us see, in (9) to (14), what importance we are inclined to assign each of these five supposed agencies.

9. *Stress.*—The relief of stress is not likely to cause a very large fraction of the softening induced by annealing; first because it is incompetent to explain the great hardening and embrittlement which gentle heating causes in cold-worked iron; and second because the progress of the soften-

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[4] *The Metallography of Steel and Cast Iron*, pp. 365 and 452 to 458. The Authors' explanation (*Bulletin* No. 109, p. 7 (January, 1916)) that the strengthening represents the removal of stress is not competent to explain either the tripling of the hardness which, as I find, mere rest after deformation may cause, or the simultaneous great loss of ductility. Thus Stromeyer (*Journal of the Iron and Steel Institute*, vol. 75, No. 3, p. 92 (1907)) finds that the ability of the outer surface of a nicked test-piece to elongate before rupture may fall from 60 per cent. to 6 per cent. on long rest, and in one case to 6 and in another even to 3 per cent. on heating to 100° for 15 min. Nor can we accept the explanation (Gulliver, *Journal of the Institute of Metals*, No. 2, p. 220 (1909)) that this hardening, strengthening, and embrittlement represent "a change from the instable Fe_3C, present in commercial steels, to the stable Fe_3C," because I find that practically carbonless steel, with only 0.01 per cent. of carbon, hardens like and perhaps even more than common mild steel of 0.147 per cent. of carbon, on heating to 100° after plastic deformation (*op. cit.*, p. 365). This strengthening and hardening which slight heating causes so markedly in iron occur only in a slight degree in brass, as is shown concordantly by the results of Bengough and the Authors (pp. 3 and 6). Indeed it is so slight as to have been overlooked by Grard (p. 3).

ing, both absolute and relative, in the Authors' results is not such as would be expected to be caused by stress-relief. Thus, on the theory that stress-relief is the chief cause of the softening, it is hard to see why a progressive increase in the degree of cold-rolling should lead either to the progressive and marked lowering of the temperature of most rapid softening, or to the marked narrowing of the temperature range in which most of this softening occurs, as recorded by the Authors (Fig. 3), or why, after 8 per cent. reduction by rolling, the softening should be only 1.4 units between 350° and 450°, yet should rise to 2.1 units between 450° and 550° and to 2.5 between 550° and 650° (Table IV).

10. *Reorientation or Disintegration.*—If we are right in conceiving that visible disintegration represents the breaking away from the initial orientation and substituting a new one in each of the several grain fragments which thereby become visible, then before any large fraction of the fragments have thus changed, the change should become visible, substituting in the earliest of the reorienting grains finer uniformly etching areas for the initial coarser ones. On this conception, disintegration is not likely to be the chief cause of the softening, because after the Authors' 4 and 12 per cent. reductions more than one-third of the total softening occurs from 100° to 200° before the reorientation becomes visible (Table IV).

Again, it is hard to reconcile the belief that most of the softening is due to reorientation with the fact that, after their 35 per cent. reduction, 80 per cent. of the total softening occurs after reorientation has become visible, and successive softenings each of about 10 per cent. of the total are found at temperatures, 200°, 300°, and even 400° higher, though the micrographs suggest no such temperature-distribution of the reorientation.

11. *Grain Growth.*—If we conceive, as seems reasonable, that the grain fragments cannot coalesce with each other to any important degree before they reorient themselves and thereby cause visible disintegration, then the great softening which, after the 4 per cent. and 8 per cent. reductions, precedes visible reorientation, cannot be referred to grain growth. This disinclines us to refer the softening as a whole chiefly to grain growth, in spite of the very great softening that accompanies the replacement of finer by coarser-grained gold in the important experiments of Fahrenwald,[5] a softening so great that we hesitate to refer it solely to difference in grain size. The Authors clearly share this belief (p. 31).

Another indication that grain growth is not the chief cause of the softening is the discordance between the curves of grain size and of hardness in Fig. 4. To judge from Plate XI there is a great retardation of grain growth from the 350°–450° to the 450°–550° range, and Fig. 4

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[5] *Bulletin,* No. 109, p. 122 (January, 1916).

tells us that there is such a retardation from the 450°–550° to the 550°–650° range, yet in passing these ranges the softening undergoes no corresponding retardation, nor does the tensile weakening.

12. *Reabsorption of the Amorphous Sheaths.*—Thin as these sheaths seem to be, if we refer to them the great hardening that cold work is capable of giving we must infer that their total volume forms a very appreciable fraction of the whole mass, an inference which is the more probable in view of the marked changes which cold work causes in the solution pressure, density, and other physical properties of the mass as a whole.

It is reasonable to hold that this amorphous metal is harder than the crystalline, because it lacks the cleavages and partings which enfeeble every crystalline mass. Hence its progressive reabsorption on heating and rest would itself naturally cause a corresponding softening of the mass as a whole. It may be that the rate of reabsorption of this amorphous metal on heating and rest varies in such a way as to contribute greatly to the variations observed in the rate of softening of the whole mass.

13. *Change of Condition of the Amorphous Sheaths or of the Crystalline Fragments which they Inclose.*—The great hardening, strengthening, and embrittlement which pure iron undergoes on rest or on slight heating after cold deformation are not readily referred to any of the four agencies which we have just considered in (9) to (12). In order to explain them I attempt to refer them to unknown changes in the nature of either the amorphous sheaths or crystalline fragments which they inclose, or both. If we grant that either or both these agencies are capable of causing these very striking changes, we naturally infer that they may also contribute to a very important degree to the softening which follows this hardening in iron on higher heating, and occurs without prominent hardening in the other metals.

14. *To sum up,* stress-relief and grain growth are not likely to contribute greatly to the softening caused by heating overstrained metal. Reorientation may contribute greatly, but it is not likely to be the chief cause. The reabsorption of the amorphous sheaths inclosing the grain fragments, and a progressive change in the condition of those sheaths of the crystalline metal in those fragments, are possible important causes. Beyond this preliminary speculation the present data do not justify our going.

It is probable that at least two agencies are at work, with effects which differ greatly, because, in the case of iron, rest and gentle heating, while effacing one of the effects of the cold work, the etch bands, simultaneously increase another and simultaneous effect of that same cold work, the hardening which it causes.

15. *Correspondence between the Authors' Conjectured Progress of*

Recrystallization and Their Observed Progress of Softening.—First we must understand the Authors' term "fragmental resolution," for which I venture to suggest "limiting grain size." They hold that, for each temperature, there is a certain " limiting grain size," "limiting" in the sense that all grain or grain fragments which are smaller than it start to grow toward it, but cannot grow beyond it, nor can grains of this or any greater size grow at this temperature. As far as I understand "fragmental resolution," it means this "limiting grain size," a term which I offer reluctantly and only because I find "fragmental resolution" confusing.

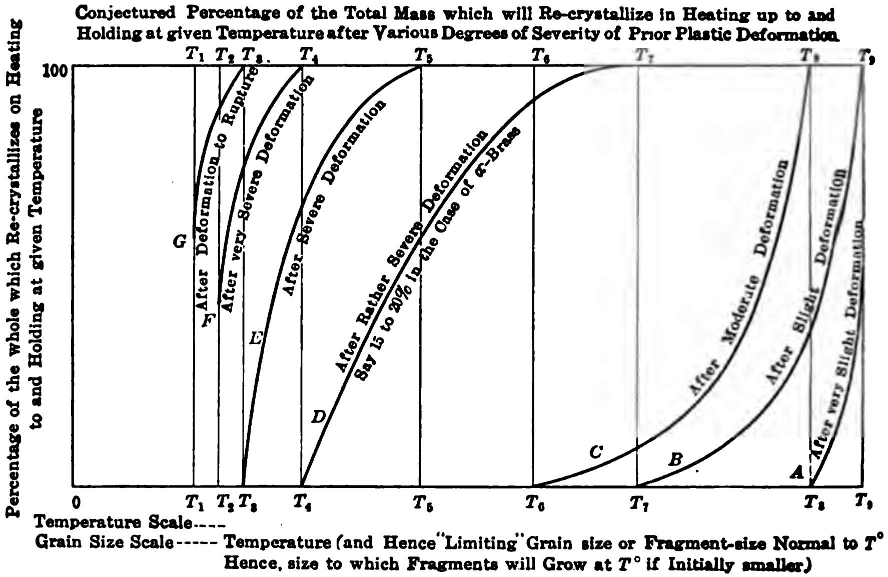

FIG. *A*.—CONJECTURED PROGRESS OF RECRYSTALLIZATION OF COLD-WORKED BRASS ON PROGRESSIVE HEATING.

The Authors give in their Fig. 2 a most interesting generalization from their hypothesis. This, their conjecture as to the progress of recrystallization, that is of the return to the crystalline state of the metal made amorphous by the deformation, may perhaps be grasped more easily as relettered in my Fig. *A*. Here ordinates show what percentage of that part of the metal which has thus been made amorphous can recrystallize in rising to given temperature and staying there 30 min. Abscissæ measure both temperature and the limiting grain size for given temperature. Far from attempting quantitative accuracy, this sketch tries only to show the general family to which the curves are expected to belong.

According to this forecast, the greater the previous reduction by cold-rolling has been, (1) the lower is the temperature at which the curves leave the horizontal axis, that is the temperature at which recrystallization starts, and (2) the steeper is the beginning of the curve. Further (3), whereas after severe deformation the curves are convex upward, after slight deformation they are concave upward. Hence the curves for great reductions form a sheaf of which the top fans out to the right, whereas those of slight deformation form a sheaf of which the bottom fans out to the left.

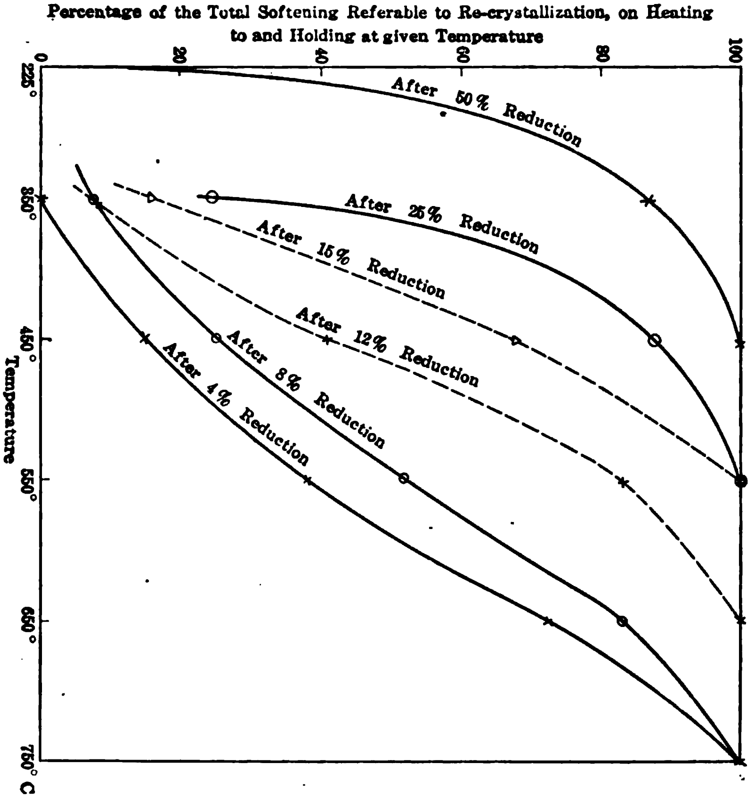

FIG. B.—OBSERVED PROGRESS OF SOFTENING OF COLD-ROLLED BRASS.

We have no data showing the true progress of this recrystallization, and hence we cannot check their forecast as to the relation of this progress to the temperature of annealing. But in my Fig. *B*, I attempt to check it indirectly by showing how the progress of that fraction of the softening which may be referred to recrystallization is related to the annealing temperature. Any such attempt must be very rough, because we do not know what fraction of the softening is due to recrystallization and what to grain growth. In view of the relatively small effect of grain size as such on hardness, I assume that the greater part of the softening is due to recrystallization, and for the purposes of this rough check I assume that the temperatures given in line 2 of my Table *A* are those at which recrystallization becomes complete, on the ground that these are the ones at which the softening is nearly complete.

Because the heatings underlying their Table IV were of 30 min., because a temperature of 255° was needed in their Table II to cause an appreciable softening after 40 per cent. reduction in about this time; and because 30 min. at 200° caused a slight hardening after this reduction; I set $T°_1$, the temperature needed to cause the first softening after 50 per cent. reduction, at 255°.

The curves of Fig. *B*, plotted on these assumptions, tend to support the second and third of the Authors' predictions, that the steepness of the beginning of the curve increases with the previous reduction, and that the curves for great reduction are convex upward whereas those for light reductions are concave upward.

Their first contention, that the temperature at which recrystallization starts is the lower the greater the deformation has been, has such good direct experimental support that it is to be accepted in spite of its not being clearly indicated by these curves. Note, for instance, that the shape of the 8 and 12 per cent. reduction curves suggests that they leave the horizontal axis at temperatures rather below than above those of the 15 and 25 per cent. curves.

16. *Is There an Equilibrium Grain Size for Each Temperature?*—The Authors' assertion to this effect (p. 18) may well be taken rather cautiously in our present ignorance as to the mechanism of grain growth. That this growth accelerates with rise of temperature, and retards itself as it proceeds, seems clear; but that it ceases completely, so that the agencies resisting coalescence come in time to arrest it completely, at relatively high temperatures, has not yet been shown, I believe. A marked retardation as the process proceeds would naturally occur on almost any theory of grain growth, but that need not imply arrest. If it is true as reported that meteorites often consist of a single enormous grain, that would indicate that, at least in this case, if time is almost unlimited coalescence becomes almost unlimited, and that here the anti-coalescence agencies never become strong enough to cause complete arrest.

17. *Certain Laws of Annealing for Alpha Brass.*—The Authors' data point to the following laws:

(*A*) As the degree of previous deformation (cold-rolling) increases from a reduction in thickness of 4 per cent. to one of 40 or 50 per cent.,[6] (1) the temperature at which in subsequent heating reorientation becomes visible under a magnification of 85 diameters falls progressively from 650° to between 275° and 300° (Table III); (2) the temperature of most rapid softening falls progressively from between 550° and 650° to below 350° (Table IV); and (3) the range of temperature in which the greater part of the softening occurs becomes progressively narrower (Table IV).[7]

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[6] Both the 40 per cent. maximum reduction of Table III and the 50 per cent. reduction of Table IV and Plate XI are correct.

(*B*) From this it follows that the annealing temperature needed to cause nearly complete softening is the lower the greater the previous reduction has been, and the longer the annealing period. For a 30-min. annealing the relation is about as follows:

TABLE *A*

1. Percentage of reduction by cold-rolling.....	4	8	12	15	25	50
2. Temperature needed for nearly complete softening, degrees Centigrade..............	750	750	650	550	550	450
3. Percentage of total softening at that temperature..............................	90.4	98.8	90.0	90.6	79.6	79.2
4. Lowest temperature of visible disintegration, degrees Centigrade.......................	650	550	550	450	350	350

(*C*) The temperature-range of most rapid softening merges into a higher range of progressively retarded further softening, which continues in every case even to beyond 750°, suggesting that the softening is due to at least two superposed changes, of which one, probably recrystallization, occurs rapidly at a relatively low temperature—the lower and narrower the greater has been the cold-rolling—and the other, probably grain growth, is spread out.

(*D*) *Decrease on Heating of the Excess of Hardness Caused by Greater Deformation.*—Here we may leave out of account the 2 per cent. reduction, because its effects are so slight compared with the accuracy of the Shore method, and the 50 per cent. reduction because this refers to a different alloy.

The excess of hardness caused by 25 per cent. reduction over that caused by 4 per cent., which is 14.7 Shore units before annealing, falls to 10.9 units on annealing at 350° and to 1.7 units on annealing at 450°. We may question whether this excess remains measurably great after annealing at 550° or any higher temperature, for though, if we consider the extreme cases, the 4 and the 25 per cent. reductions, a slight excess persists even on heating to 800°, as shown in line 4 of Table *B*, yet for

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[7] This agrees with Muir's observation that the greater the hardening caused by overstrain, as shown by the rise of the yield point in steel, the lower is the temperature at which softening sets in (*Philosophical Transactions of the Royal Society of London*, vol. 193 A, p. 22, 1900), and with Chappell's that the greater the deformation the lower the temperature at which the recrystallization (disintegration and subsequent coarsening) of overstrained steel occurs (*Journal of the Iron and Steel Institute*, vol. 89, pp. 471, 496, 1914).

TABLE *B*

1. Annealing temperature, degrees C....	Unannealed	350	450	550	650	750	800
2. Hardness after 25 per cent. reduction..	27.4	23.6	13.7	11.8	10.2	8.2	7.8
3. Hardness after 4 per cent. reduction...	12.7	12.7	12.0	10.9	9.3	8.0	7.5
4. Difference, Shore units[8].............	14.7	10.9	1.7	0.9	0.9	0.2	0.3

temperatures above 450° this excess hardly exceeds the limits of error. Moreover, if we compare all five of the reductions, we find great anomalies. For instance, the specimens reduced by 4 per cent. are actually reported as harder than those reduced by either 8 or 12 per cent., after the 650°, 750°, and 800° annealings severally.

Again, for each of the four annealing temperatures, 550°, 650°, 750°, and 800°, there are five reductions that we are considering, those of 4, 8, 12, 15 and 25 per cent. If for each temperature we compare the hardness after any one reduction with that after the next greater reduction, we have four such comparisons at each temperature, or 16 for the four temperatures. Though in eight of these 16 cases the hardness given by the greater reduction exceeds that given by the less, yet in seven cases the reverse is true, and in the remaining case the greater and the less reduction give the same hardness.

(*E*) *The Temperature at which Disintegration First Becomes Visible may Suffice to Efface the Excess of Hardness Given by Greater Reduction.*— This refers to the temperature at which the reorientation becomes visible under a magnification of 85 diameters. For instance, this temperature is 550° for the reductions of 8 and of 12 per cent., and at this temperature the residual hardness is actually greater after the 8 than after the 12 per cent. reduction, though the difference hardly exceeds the limits of observational error.

(*F*) As the temperature rises, the softening accelerates greatly and continuously up to a maximum rate. For instance, a softening amounting to 3 Shore units is made 6.4 times as rapid by a 50° rise, from 225° to 275°, and about 1,000 times as rapid by a 100° rise from 225° to 325° (Table II, p. 7).

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[8] By a "Shore unit" is meant a difference of 1 in the unit place of the Shore scale, hardness 14.3 being 2 units greater than 12.3.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

W. H. BLAUVELT, Syracuse, N. Y. (communication to the Secretary*).—I have read Mr. Howland's paper on Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces with much interest, and believe that his method of analysis of the subject will be of material assistance in our study of the action of coke in the blast furnace.

Mr. Howland quotes Professor Richards' statement that from the standpoint of the generation of the maximum quantity of heat in the furnace, Grüner was right in saying that all the carbon burned in the furnace should be first oxidized at the tuyères to CO and all reduction of oxides above the tuyères should be caused by CO, which thus becomes CO_2. It is clear that the maximum quantity of heat is generated in the furnace when the greatest amount of carbon is burned to CO_2 irrespective of whether it is burned at the tuyères or further up the furnace. So it would appear that Grüner's theory is sound regarding the maximum production of heat only because if all the carbon is burned to CO at the tuyères the reduction of the ores will have to be effected by the further oxidation to CO_2. Whereas, if the reduction of the ores is effected in part by solid carbon, part of this carbon might be oxidized only to CO, resulting in a higher percentage of CO in the escaping gases than from Grüner's "ideal conditions." But in studying the combustion of coke in the furnace, it is clear that the production of the maximum quantity of heat is not of the first importance in blast-furnace operation, or in the utilization of the fuel charged into the furnace. To my mind the production of a high thermal head at the tuyères is of the first importance, and the best coke is that which reaches the tuyères in proper condition to produce the highest temperature at the tuyères, and in just sufficient quantity to do the amount of work required there under the conditions produced by this maximum temperature. The combustion of a much larger amount of fuel at the tuyères, under conditions that will fall short of producing the highest possible temperature, cannot produce as good results, either in fuel economy or output. If this statement is correct, does it not correspond with Mr. Howland's statement that too much wind is inimical to low coke consumption? Nothing is more fatal to obtaining the highest temperatures than an excess of air for combustion. In the blast furnace an excess of air dilutes and cools the products of combus-

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* Received June 30, 1916.

tion, reducing the maximum thermal head at the tuyères, and the larger volume carries the high-temperature zone too high in the furnace.

There has been a great deal of discussion as to what are the qualities which make the best furnace coke. The wider use of the byproduct oven has brought the control of the coke closer to the furnace man, and makes it possible to modify its structure in many ways, so that the question "what are the best specifications?" has become a very live one. It will probably be generally admitted that furnace coke should be of nearly uniform size, and many furnace managers are eliminating all coke below ¾ in. and above 4 or 4½ in. Also that the best coke is that which is sufficiently strong to resist undue abrasion and crumbling by attrition with the stock, and of an open porous structure that will permit the most rapid combustion when it reaches the tuyères. Many large users agree that the coke should never be overcoked, beyond the point of producing a sufficiently strong structure, as overcoking quickly reduces the combustibility.

If Grüner's ideal gives the best furnace operation, we should want a coke that is resistant to the oxygen in the ore but easily combustible at the tuyères, which is a contradiction of qualities. If my argument is correct, that the furnace man wants the greatest thermal head at the tuyères rather than the production of the greatest quantity of heat in the furnace as a whole, then he is willing to sacrifice some coke by solution in the oxidizing gases in the upper part of the furnace, provided he can obtain a sufficient quantity of coke at the tuyères, of a quality that will permit rapid combustion with the minimum amount of air, thereby giving him the maximum thermal head.

The Rifling of Diamond-Drill Cores

Discussion of the paper of Walter R. Crane, printed in *Bulletin* No. 113, May, 1916, pp. 823 to 833.

H. M. Roberts, Minneapolis, Minn. (communication to the Secretary*).

—The rifling of drill cores is a frequent source of interest to men engaged in diamond drilling. Previous to the appearance of Dr. Crane's paper, there has been little record of connected observation or rigid speculation as applied to the cause of rifling. Replying to one of Dr. Crane's letters of inquiry, I ventured my opinion that the cause of rifling was a complex problem in physics, which I had not attempted to solve up to that time, except to ascribe it to the general cause of rotation and vibration of the rods. The question is worth attention not only as a matter of curious scientific interest but also for the reason that it stimulates thought on the mechanical action which takes place at the end of a diamond bit. In discussing Dr. Crane's paper, I still recognize the complexity of the problem and merely note a few observations of fact with the inferences which may be drawn from them. I am indebted for suggestions from W. J. Mead, F. F. Fredlund and James W. Hunter among others on the staff of the E. J. Longyear Co.

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* Received June 24, 1916.

It seems clear that the cutting medium which causes uniform threading must be the diamond bit itself. The deep, regular rifling of such dense rocks as granite and norite for long intervals admits of no other reasonable assumption.

Rifling occurs in many different rocks, both hard and soft, but in all instances noted there is one feature in common: the rock is homogeneous over the extent of the rifling.

Wall of Drill Hole Rifled

The wall of the drill hole itself is rifled, as well as the core. This has been observed in the shaft of the Isabella Mine in Northern Michigan, which was sunk on a 2-in. hole in diabase.

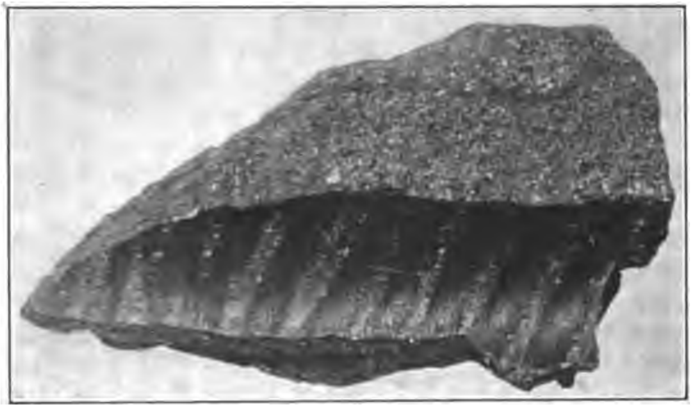

Fig. 1.—Specimen from Shaft of Isabella Mine, Cascade Range, Mich., Showing Rifling on the Wall of an "N" Drill Hole.

Fig. 1 shows a specimen taken from the walls of this hole. It is reasonable to suppose, therefore, that rifling of the core is also accompanied in most instances by rifling of the wall of the hole. This leads to the inference that perhaps the immediate end of the bit is responsible.

The Diamond Bit

A consideration of the great pressures which bear upon a bit during the drilling makes it quite certain that all the stones work and engage the rock together. Examination of a worn bit shows that all the stones not as great as might be supposed, for when a bit is blocked under great pressure, and the engine is stopped and released, the chuck seldom revolves back more than a turn or a turn and a half. During this process the bit is presumably fast on bottom.

As the rods revolve, transverse waves advance down the rods at every azimuth, like waves of light. That waves of this type are produced is shown by the whipping out of uncased holes when drilling soft iron formation. The samples are often vitiated in this manner. Anyone who will climb the tripod and look down on the water-swivel when the rods are vibrating will see there a distorted but nevertheless fairly true picture of what is taking place at the bit on the other end of the rods. It is possible to count distinct beats in the play of the swivel to and fro as the rods vibrate.

The regularity of the grooving in the threaded core indicates that the waves producing it have a definite time interval with a fixed relation to the revolution of the rod. The fact that all the stones are working at the same time does not present any particular difficulty when it is remembered that in any one instance of threading the stones are always at fixed intervals with respect to the axis of the bit and therefore have a fixed relation to any wave that affects the bit and a fixed relation to the period of revolution. In the instance cited by Dr. Crane where pentagonal core is formed, the time of vibration must be nearly a definite divisor of the time of revolution. That is, during one revolution of the bit there were approximately five major wave motions which produced a portion of five threads on the core for a longitudinal distance of less than $\frac{1}{100}$ in. In all other instances where the pitch is flatter the ratio must be more intricate. When right-handed threads are produced, the period of vibration is incommensurable and in this instance thereby deferred slightly each time with respect to the core as the rod revolves. When left-handed threads are produced, the period of vibration is also incommensurable, but in this case occurs just an instant earlier with respect to the core during each succeeding revolution.

Overtones

In wave motions of this kind it is conceivable that there are overtones; that is, if the period of one wave length be T, then there are wave lengths of $\frac{1}{2}$ T, $\frac{1}{3}$ T, etc. The braided threads on many pieces of core are suggestive of secondary action of this kind (see Fig. 2 of Dr. Crane's paper). All of the laws of resonance and interference in wave motion would enter into the amount of play in the bit as it revolves. Again, when relatively smooth core is produced, it is probably the result of the impact of countless numbers of periodic waves offsetting each other.

There are many other apparently simple phenomena in nature which are due to a combination of periodicities, as a beam of light or the general strike of a complexly folded iron formation.

The net result of the wave motions down the rods operating under the laws of resonance is translated into a wave whose length is nearly parallel to the circumference of the core and whose amplitude is along the radius, thus producing uniform threading. This is recorded in the rock like the mark on the indicator card of an engine. Only in rare instances do all the factors combine under the laws of resonance so that a deep definite tracing of the wave motion is formed.

Mathematical Interpretation

We may attempt to arrive at a rough mathematical interpretation of wave motion at the end of a diamond bit as expressed by threads in the core.

Let R = the number of revolutions of the bit during any interval of advance.

N = the number of threads appearing on the cross-section of core

$F(p)$ = some continuous or discontinuous function of the pitch of the threads, p having values from negative to positive infinity, $F(p)$ approaching the values -1 and $+1$ at these ends, and approaching the value zero as p approaches zero, the value of p being related to the value of N.

Let W = the number of major wave motions during any interval of advance of the bit, as indicated by the threads.

Then $W = R\,N\,F(p)$

Thus in the instance where pentagonal core is formed, and N equals 5, p approaches infinity and $F(p)$ approaches the value 1, if it be assumed that 600 revolutions of the bit have been made during the delivery of 1 in. of core; then $W = 5 \times 600$ or 3,000, which is indicative of the number of major vibrations of the bit with respect to the axis of the core during its spiral advance of 1 in.

When the pitch p is left-handed, $F(p)$ may be considered as varying from 0 to -1; when the pitch is right-handed $F(p)$ varies from 0 to $+1$. As p becomes small in either case the value of N increases and W increases approaching the case where many fine threads of low pitch are produced; W increases greatly, which is to say that a cylindrical core is formed when there are a large number of exceedingly small wave motions of the bit to and fro. Considering an instance when N remains constant and the threads ultimately coalesce while "p" decreases in value, then the value of W decreases. The ultimate end of this process is the production of a smooth core with slight wave motion of the bit.

Inspection of this equation shows that as R increases, W must increase.

This expresses the violence of vibration when harmonic motion of the resonant type is set up during a high speed of revolution.

This mathematical statement is merely an attempt to express some of the relationships which appear, by speculating on the nature of the phenómenon. It is doubtful whether it is possible to determine rigidly the value of $F\ (p)$ in any instance or to tell why p should be left-handed or right-handed or to devise any means of telling why N is 5 in one case or 9 in another. The strength of the rods, the number and character of the carbon, the way they are set with respect to each other, the water pressure in the hole as determined by the force of the pump, the clearance of the bit, etc., the amount and character of the deformation in the metal of the bit at any particular time, the kind of drilling machine, and the manner of its set up, the depth drilled, the nature of the rock, the presence of soft horizons above in the hole which may affect the vibration of the rods, the speed of rotation and the pressure on the rods—the factors which influence these values are numerous and therefore difficult to analyze into their true proportions.

Practical Considerations

Dr. Crane's practical application of the results of his research is discouraging in these days of keen competition for drilling contracts. I refer to the suggestion that it is possible to reduce "the vibratory action of the drilling mechanism by reducing the speed of rotation of the rods." An old drillman in commenting on this says that he can break up the vibration by increasing the speed of rotation. This suggestion might better be stated thus: The violence of vibration can be broken up by varying the speed of rotation.

This destroys one of the constant conditions essential to harmonic motion. The greasing of the rods dampens the wave and sets up interference by releasing friction at antinodes of the waves, thus destroying another constant. This may be inferred by examining the line of greased rods when pulled from the hole and by observing that the grease is worn clean at definite intervals.

In making this comment on Dr. Crane's conclusion I would not be understood as casting doubt on the utility of this type of research. An analysis of all the conditions which govern the advance of a diamond bit might enable operators to control the direction of diamond-drill holes. At present the hole usually takes its own direction.

Some practical considerations have occurred to me while dealing with the subject. They are no doubt commonplace to many old drillmen, but seem worth recording: When drilling deep holes in homogeneous formations, it will reduce the carbon wear and increase the footage to use stiff new rods and to use a blank bit which has the smallest possible excess in diameter over the size of the rods. The use of eight stones in a bit rather than four or six will reduce the possibility of one stone carrying the burden of the work. The bit should be reset frequently so that deformation of the weakened steel in long-used bits will be avoided; thus the gage of the hole may be kept more accurately. These precautions will set up conditions adverse to the development of resonant wave motion to and fro in a bit and thus tend to keep it working straight ahead, which is the true business of a diamond bit.

The Emerald Deposits of Muzo, Colombia

Discussion of the paper of JOSEPH E. POGUE, printed in *Bulletin* No. 113, May, 1916, pp. 799 to 822.

EDGAR T. WHERRY,* Washington, D. C. (communication to the Secretary†).—Dr. Pogue's presentation of the facts concerning the emerald deposits is very clear and convincing, and the only addition that I can suggest is a summary of previous theories of origin. He makes it evident that the pegmatite theory is the only one capable of explaining the existing relations, but upon certain details there may be some difference of opinion. If I understand the term pneumatolytic, it does not imply that all the elements concerned in a given deposit were transported as gases, but rather that the crystallization of these elements into the various minerals was favored by the presence of certain gaseous substances, notably H_2O, CO_2 and HF. It is highly improbable that the oxides of glucinum, aluminum, chromium, and silicon, which enter into the composition of the mineral emerald could have been transported in the gaseous form. The same is true of the metallic constituents of the parisite and other associated minerals. The explanation suggested, that solutions separated into liquid and gaseous portions, the latter ascending and forming the emerald in the upper portions of the rock only, therefore, seems to me untenable.

When two formations exist side by side and one, *A*, is mineralized while the other, *B*, is barren, the possible explanations may be classed as (1) chemical, and (2) physical.

1. Some chemical feature of *A* not found in *B* might have caused crystallization of certain minerals in the former, which did not appear in the latter. In the present instance both rocks appear to be so similar chemically that no such effect can be looked for. Pogue mentions carbon as a possible precipitating agent, but describes both formations as carbonaceous, so that the difference in the minerals of the two can not be thus explained.

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* Assistant Curator, Division of Mineralogy and Petrology, U. S. National Museum.

† Received June 28, 1916.

2. The physical condition of *A* might have permitted or encouraged the passage of the solutions, while that of *B* retarded or prevented it. In the present deposit some mineralization occurs in both formations, calcite veins, albite, and pyrite being found in both, whereas emerald, parisite, and a number of minerals of minor importance occur only in the upper, *A*. It seems to me that this difference may have been produced by a change in the composition of the solutions during the progress of mineralization; at first these brought in only the constituents of albite and pyrite and deposited them with calcite dissolved from the wall rock, in both formations; the openings in *B* became completely filled, while the more numerous or larger ones which would naturally have developed in *A*, since it was the uppermost formation, remained partially open. Then, when during later phases of the mineralizing activity the constituents of emerald and parisite appeared, they were deposited only in *A* because *B* had become impermeable.

History of the Flotation Process at Inspiration

Discussion of the paper of RUDOLF GAHL, printed in *Bulletin* No. 117, September, 1916, pp. 1627 to 1681.

DAVID COLE, EL PASO, TEXAS (communication to the Secretary*).—I have read with great interest Dr. Gahl's painstaking paper giving us the details of development of flotation at Inspiration, and it seems to me that he has covered the ground so completely that there is little to discuss or criticize in connection with his subject. While it was in operation, the test mill in which the campaign was carried on was the Mecca of mill men and metallurgists. There was much to interest students of milling methods in addition to the flotation experiments, and readers of Dr. Gahl's paper will miss the information they hoped to get when the "pilot" mill results were finally compiled, and I hope that the following remarks outlining from a distance some of the things that happened there will bring forth the rest of the story of the performances recorded in these important experiments.

Prior to the use of flotation methods in concentration it had long been recognized that the "unavoidable" losses of sulphides were in the slime which is inevitably produced in the grinding operations required to free the minerals to be separated. Classifying the feed prior to its final stage of treatment had long been in style in milling. This assisted the sand-handling machines and resulted in lowering the tailings made by them, but at the expense of the slime-handling department; and while little was expected of the latter, the wisdom of complicating the process by hydraulic classification was being seriously questioned. Indeed, at the time that the gravity method flowsheets were being developed for Inspiration ores, there was a minor revolution in milling methods impending involving the elimination of the hydraulic classifier. This change gave no promise of higher extractions but did promise greater simplicity and much less floor space per unit of capacity. Such mills promised to be less costly to build, less expensive to operate, and equally efficient. Wholesale concentration in relatively small space would get as good results as piecemeal concentration had been getting in the multiple operations of the spreadout plants into which the porphyry treatment mills had degenerated.

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* Received July 8, 1916.

The one great drawback was the slime. The desiderata of the engineer and the manufacturer was, on the one hand, to provide grinders which would produce the minimum of slime, and, on the other hand, improved "slimers" which would give the maximum recovery from the finest products. The technical press of the period reflected this state of things in the advertisements of those having the latest novelties to sell. There was much revamping of old ideas with some refinement, but nothing new. Tube-mill grinders were "taboo" for concentration, because they had a bad reputation as slimers. Automatic canvas plants were being exploited and very ingenious multiple-deck table devices were being offered as the remedy—the only possible remedy.

Looking backward no further than 1912, when Inspiration began to study its milling problems, we now see that we were without effective resource in treating real sulphide slimes. Mr. Callow's investigations and experiments had apparently demonstrated that some departure from the usual gravity practice would be advantageous and that a high recovery for that method would be possible on the granular material, but he threw small light on slime treatment. This was the situation when the Inspiration company was endeavoring to determine its mill treatment scheme.

After reviewing from every angle the results of experiments on the ore and other information available, a modified flowsheet was finally crystallized by Mr. Burch and adopted by the management. Mill plans were drawn for what was to be a most highly developed 7,000 tons per day gravity process plant, and work was immediately inaugurated to carry out these plans. The mill site was selected, and much active work had been prosecuted before flotation (by this time being hastily tried in the old experimental plant) had so far won its battle that results could be viewed otherwise than probably too good to be true.

The apparent promises of flotation were extremely attractive, because the process would be simple and would solve the all-important slime problem. The process would have a greatly reduced number of stages; the plant would be much smaller per unit of capacity; the cost of construction per unit of capacity would probably be very much less; the use of water would be minimized; grinding would have to be carried further than usual, and would be the main item of milling cost, but this was not a very great handicap because sliming did not matter. What system of grinding would be best to use and what is the best machine? Would it be possible to parallel the small test-mill metallurgical results on a full-tonnage basis, and finally would it not be too hazardous to accept so revolutionary a process with so much experiment in its makeup?

The mine preparation program which had been settled upon would produce about 600 tons daily of freshly broken ore directly from the headings. This happened to be the rated capacity of a full-sized Minerals Separation unit which the flotation people were urging as a means of improving their extraction. If this ore from the headings were put in stock pile in the usual way, pending the completion of the milling plant, the ore would oxidize to some extent and besides would involve reclaiming expense later on. Why not mill it all as fast as produced, in a real "pilot" mill, wherein not only flotation problems, but grinding problems, power consumption, use of water, preparation of sticky concentrates, and other vital questions which might come up could be definitely threshed out under what would be regular commercial conditions on full-sized machines? In this size of "pilot" operations, labor would be used economically and production would almost, if not quite, pay all of the current expenses, except the mining cost.

This program was adopted. That the decision to carry it out was a wise one is shown in Dr. Gahl's paper, and that it paid its way is shown by Mr. Mills' annual report for 1915, in which he says: "Contrary to the usual experience, this test mill paid the cost of its construction, its operation expense, the present average mining cost on ores treated, and something besides, and has been written off the books."

Anybody with a real idea applicable to the problem could get a hearing and a tryout in this extremely practical commercial sized laboratory where sampling was in the hands of engineers and the results were compiled in a way to make them comparative and valuable. Thousands of dollars were spent by the company, by inventors, and manufacturers in demonstrations. Expense was subordinated. Heavy shipments by express were made when necessary to hasten the work, and much more than flotation was developed in this "pilot" plant.

Four different types of Symons crushers and pulverizers were tried. Three of these were marked and very interesting departures from ordinary practice. The company had purchased the Hardinge mill patent rights for Arizona and four forms of this mill were installed to determine the best to use, and these were kept busy during nearly the whole campaign. A long parallel tube mill was installed and run in competition with the Hardinge mill. A high-speed Huntington-type grinder at one time attracted much interest. Hammer pulverizers of two different makes were also tested. Various linings and grades of flints were tried in the pebble mills. Steel balls in place of pebbles were advocated and a carload purchased.

In the latter part of the testing period the Marcy type of ball mill, especially designed for using iron balls larger than usual, and adapted to crush from breaker sizes to 48-mesh in closed circuit in one operation, was installed and perfected. This grinder proved capable of a greater range of reduction than had previously been thought possible, from an initial feed as coarse as 3-in. cubes. It is a ball mill pure and simple, having large capacity in small space. It makes use of a perforated diaphragm to keep the balls and charge inside of the mill until the latter will pass a $\frac{3}{16}$-in. opening, and it has the equivalent of a peripheral discharge. An overflow classifier determines the finished size and the oversize is continually returned to the grinding chamber. This mill uses little water in the grinding chamber, so that its charge of ore is mortar like in consistency. It was quite successful. There was nothing in the Hardinge equipment to parallel it because the Hardinge mills were built for pebble mills and did not have feed scoops or openings adapted to handle as coarse a feed, and the linings would not stand up under the ball load. Would the Hardinge machines when built as a ball mill with the required strength, type of lining, size of feed and discharge openings, do as well? To wait for a mill to be made over or a new one manufactured would take too long, so the Marcy type was adopted and, contrary to what I think is the popular impression, the conical-type steel-ball mill *vs.* the Marcy-type steel-ball mill did not receive a tryout.

Recording electric instruments were installed so that accurate power records could be continually made while the various machines were being operated.

Several varieties of drag and rake classifiers and two types of vacuum filters were installed and records made. The efficacy of high-reduction herringbone gears driving ball and tube mills became a matter of interest on account of the troubles that developed in them, and the reasons for these troubles, which would make a story by itself when discovered.

This testing work grew into a process of elimination and the scrap pile grew steadily. Some blasted hopes may be buried there, but it does not follow that all of the machines or materials that were returned to the sponser or that found their way to the scrap heap were entirely unfit. It was necessary to choose and to discard, and that there is no acrimony in connection with the matter speaks well for the type of justice and judgment that prevailed. Doubtless the use of some discarded things might have answered as well, but it would be hazardous indeed to say that anything vital on the score of cost or recovery of values failed to receive recognition in the final selection. One of my mental offspring was among the fallen. It held out for a long time and I greatly appreciate the favorable mention which Dr. Gahl has made of it. I would have been pleased to pursue its development further, but I can not at this time see where its adoption would have saved any more copper or any more money in getting the copper than the ones that were selected, and I think this will apply to all the "late lamented" concerned in the campaign that was carried on. In other places and on other ores the conditions would not be the same. The ratio of concentration, or the crushability factors particularly, may so change conditions as to indicate the use of ideas that were discarded at the Inspiration test mill.

When the comprehensive plan of testing on a large scale was decided upon, I think Dr. Ricketts and Mr. Mills recognized that the records made might be extremely valuable to engineers and metallurgists, and I believe the matter of writing the story of the campaign was considered and that records were started with that ultimate end in view. Experience added to theory is more valuable than theory alone; the scrap heap more eloquent than the machine promotor, and the wornout unit in the "bone yard" is often more interesting than the one shown on the original blue print. It is seldom indeed that experiments are possible under such auspices and on such an extensive scale as those carried out in the Inspiration "pilot" mill, and I hope that Mr. Burch or Dr. Gahl, or both of them, will, with the approval of the Inspiration company, find time and inclination at an early date to prepare a paper or papers which will extend, through the *Transactions*, the value of these experiences to our members.

Referring to the final flowsheet adopted, I note that hydraulic classification had no place in the 600-ton "pilot" mill experiments which Dr. Gahl has so interestingly described, and I note that he has not referred to the reason for retaining this remnant of the old system of concentration in the flow sheet.

I note that 3 tons of water per ton of ore handled is required in the flotation operation and that 3 tons more is added in the subsequent table treatment which, of course, includes the hydraulic classification operation practiced, and I presume that something more than one-half of the last 3 tons is added in the classifiers themselves; and since the water is clarified and returned, the addition of unnecessary water would entail expense for clarification and pumping which would not be required if less water were used.

In my work in the concentration of ores I have not been able to become very enthusiastic over hydraulic classifiers, and since flotation has come to the rescue of the slimed sulphide I find myself less enthusiastic than ever over their use.

When I was a lad in the Black Hills I was fascinated in watching a certain mountain spring in which polished micaceous particles glistening in the sunlight would be caught in the current rising from an orifice in the bottom of the sand funnel and be flirted to the surface, sail across the crater and fall upon the conical sides to be again methodically returned over the same route. It was interesting to watch the disturbance caused by dropping a handful of foreign sand and silt into the funnel and have it "classified" and washed clean, a new form of crater finally being established with the changes of average sizes retained. My first contact with hydraulic sizing in concentration was studied from that foundation. The spring took its time to do a good job; it worked the charge over repeatedly. All the silt went out quickly and the fines gradually went overboard with a rapidly decreasing ratio, until the crater would settle down again to its regular work of turning the mobile contents over and over in a new condition of equilibrium. But I soon learned that the beauty was all taken out of the process in its commercial application.

The process witnessed in the action of the spring was balanced, precise, and definite, and quite at variance with what we witness in watching through glass the operations going on in a "teeter chamber" of the metallurgical hydraulic classifier, which seems to me to be of little value except in its office of washing out the slimes which used to be the main source of the loss in treating unclassified material upon concentrating tables.

Following the thorough combing out of the slimed values as effected by the splendid flotation treatment that the pulp has previously received in the Inspiration final flowsheet, I am inclined to question the value of the subsequent classification by hydraulic means. I note that the flotation tailing is split into slime and sand at the drag belts, that there is very little slime left in the sand portion, and that what little there is (on account of the previous frothing) is completely devoid of value which the tables can save, as indicated by the fact that the main slime overflow of the drag-belt separators is discharged to waste without further treatment. It seems to me that it ought to be possible to eliminate the classifiers, and I would like to ask Dr. Gahl if there has been any trial to determine what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation.

Comparative Friction Test of Two Types of Coal-Mine Cars

Discussion of the paper of P. B. LIEBERMANN, printed in *Bulletin* No. 114, June, 1916, pp. 1057 to 1065.

EDWIN M. CHANCE, Wilkes-Barre, Pa. (communication to the Secretary*).—I have read with great interest Mr. Liebermann's paper reporting dynamometer tests of mine cars and wish to express my appreciation of this investigation. The coal-mining industry needs just such precise study of the problems with which it is confronted.

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* Received June 24, 1916.

It would seem, however, that the writer has demonstrated that a high-grade grease is superior to black oil as a mine-car lubricant rather than that roller-bearings are superior to plain. While it is reasonable to believe that roller bearings will cause less friction than plain when used on the trucks of mine cars, still this point can not be conceded on the strength of the tests quoted, for in these tests grease was the lubricant of the roller bearings, while the usual black oil, of low lubricating value, was the lubricant of the plain bearings. Under these conditions, a case is made for the lubricant rather than for the bearings.

Some time ago certain of the coal mining companies with which I am associated, called upon me to have a high-grade grease prepared for them for use in plain mine-car bearings. This problem was successfully solved and this lubricant has now been in use for some years. The fact has been established that the ordinary plain mine-car journals are especially susceptible to such a lubricant and that its use greatly decreases the drawbar pull of these cars. The fact must be borne in mind, however, that the exigencies of this service require a special grease and that the use of the ordinary grades of cup grease will surely spell disaster.

It would be indeed interesting to have tests of the same degree of precision as those reported made upon cars with plain bearings lubricated with black oil and special mine-car grease and to compare the latter with tests under like conditions made upon cars equipped with roller bearings.

CHARLES LEGRAND, Douglas, Ariz. (communication to the Secretary*).—Mr. Liebermann's paper on friction tests of two types of coal-mine cars is very interesting. It is a pity that the dynamometer car was not designed for use on tracks of 18 to 24 in., as this is the most usual gage in metal mines in this part of the country.

I have tried to make tests by means of an ordinary spring dynamometer and find, as stated by Mr. Liebermann, that this method is very unsatisfactory, as the conditions of mine tracks are too variable to produce a steady pull on the dynamometer. It would be very valuable to the mining industry to have accurate tests taken on narrow-gage track, to determine not only bearing friction, but the total track resistance. Such tests would convince the mine managers that it pays to lay heavy tracks and keep them in good condition.

Personally, I have always been in favor of the roller bearing, not only on account of its lower friction but also on account of its smaller maintenance and lubricating costs. To be successful, however, this type of bearing has to be dustproof. On a narrow-gage car this is usually done by inclosing the whole axle in a casting which also supports the two bearings. This casting acts as an oil reservoir and permits the car to be run a whole month with one oiling.

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* Received July 5, 1916.

I hope that Mr. Liebermann will continue his work and make tests of the narrow-gage cars.

John Phillips, Pittsburgh, Pa. (communication to the Secretary*).—I am familiar with the test made by the Greensburg Coal Co. and described in Mr. Liebermann's interesting paper, although I was not personally present. Mr. Liebermann's results on the Hockensmith plain-bearing wheels compare closely with tests which have been made elsewhere and show a considerable saving in favor of the roller bearings. However, as the grade increases, the ratio in favor of the roller bearings decreases until, on a 4 per cent. grade, their efficiency is only 11 per cent. greater than that of the plain bearings, as shown by the following official figures of the Hyatt company:

Test, Oct. 3, 1915

	Pounds
Weight of car	2,650
Weight of coal	4,500
Total	7,150

Weight of 20 cars loaded, 2.4 per cent. grade average	71.5 tons.
Correction	48 lb. per ton.

71.5 × 48 = 3,432 lb.

4,280	5,190
4,350	5,070
4,410	5,250
3)13,040	3)15,510
4,347 Hyatt	5,170 plain
4,347	5,170
3,432	3,432
915	1,738

Frictional resistance for Hyatt, $\frac{915}{71.5}$ = 12.8 lb. per ton.

Frictional resistance for plain bearings, $\frac{1,738}{71.5}$ = 24.3 lb. per ton.

Drawbar, Pounds per Ton		Grade, Per Cent.	Per Cent. Saving	Percentage Cars Possible to Add in Case of Roller Bearings
Roller Bearings	Plain Bearings			
12.8	24.3	level	47⅓	90.0
32.8	44.3	1	26.0	35.0
52.8	64.3	2	17.9	21.8
72.8	84.3	3	13.6	15.8
92.8	124.3	4	11.0	12.4

It will be noticed that all the readings were taken on a straight track on an average grade of 2.4 per cent. Had any of the readings been taken on a curve, we are confident that the drawbar pull on both the roller bearings and the plain bearings would have been considerably higher when the train was rounding the curve, because the axles of the Hockensmith trucks are of straight cold-rolled steel without collars, and in rounding the curve the car naturally shifts against the inside wheel. The rear hubs of the wheels are necessarily large in order to admit the bearings, and the large hub grinding against the box creates considerable friction, which would increase the drawbar pull on a curve.

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* Received July 10, 1916.

Records of three tests conducted by the Engineering Department of the Pittsburgh Coal Co., on three different types of wheels, are given below. At Essen No. 3 mine, the Jarvis Adams roller bearings were used; at the Champion mine, Eureka wheels and angle-bar trucks (the wheels being similar to the plain-bearing wheels used in the Greensburg test), and at the Dickson mine, our open-cap wheel trucks. Inasmuch as the drawbar pull on the Hockensmith plain-bearing wheels at Champion agrees very closely with the results on the Hockensmith plain bearings at Greensburg, the results may be taken as approximately correct, and show the following comparison:

Hyatt roller bearings at Greensburg	12.8 lb. per ton.
Jarvis Adams roller bearings at Essen No. 3	15.14 lb. per ton.
Open-cap trucks, Dickson mine	18.48 lb. per ton.
Hockensmith plain bearings, Champion mine	24.40 lb. per ton.
Hockensmith plain bearings, Greensburg	24.30 lb. per ton.

The figures on the Hyatt bearings were lower than we anticipated, as we had always supposed that the Jarvis roller bearings when new and in good condition were the easiest running on the road. Our open-cap truck costs about the same as the Hockensmith angle-bar truck, but runs about 25 per cent. easier. The cost of the roller-bearing truck is about $10 to $12 per car more than our truck, and on the basis of the Greensburg figures runs about 30 per cent. easier.

From the above data, I think that it is safe to say that it would be well for Mr. Liebermann, or some person equally well qualified to carry out this class of tests, to pursue it much further, both as to different kinds of wheels, and to the character of roads, which would include different grades and varied curvature, before a thorough comparison could be made and a conclusion arrived at that would be entirely satisfactory.

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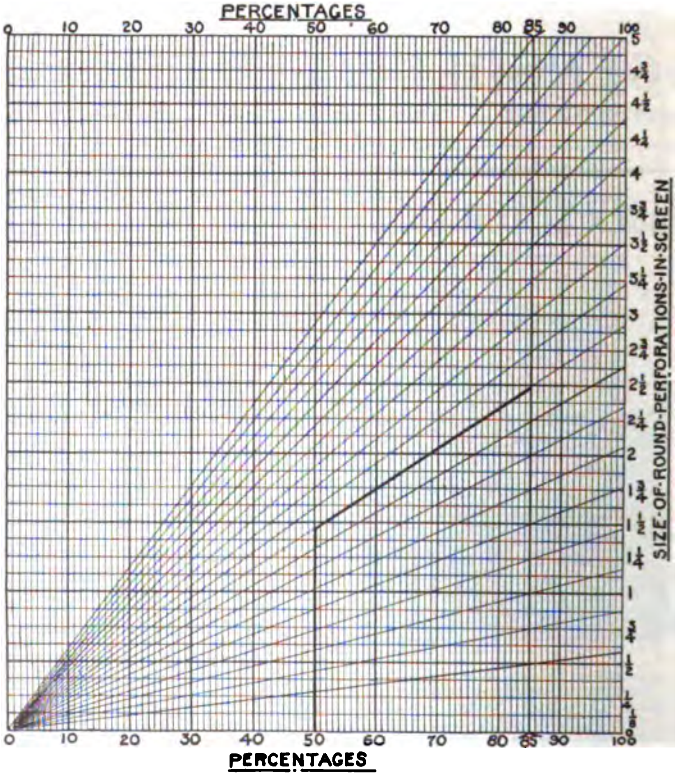
INDUSTRIAL SECTION

Some Statistics on Crushing

In Bulletin J 1, recently published by the Traylor Engineering and Manufacturing Co., it is stated that the largest jaw crusher ever built has a jaw opening of 66 in. by 86 in. and weighs 680,000 lb. It is said to be in successful operation and has a capacity of 490 tons per hour crushed to 12-in. size; 415 tons per hour crushed to 10-in. size; 330 tons per hour crushed to 8-in. size.

The same company, in its Bulletin G 3, gives a diagram showing the percentage of a crushed product which will pass through screens of different sizes of perforation from a machine adjusted to a certain size screen. This diagram is reprinted below for the information of our members and the following notes apply to it.



In order to prevent an excessive amount of fines, crushing machines are usually set so that 15 per cent. of the product will be larger than the maximum size desired, and this oversize is then recrushed by smaller

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machines. This diagram is so drawn, therefore, that the diagonal lines intersect the horizontal lines exactly on the vertical line indicating 85 per cent., and the diagonal corresponding to any given adjustment passes through the intersection of the 85 per cent. vertical with the horizontal corresponding to the same size screen.

Example.—To determine the percentage of the product from a crusher adjusted to 2½-in. screen which will be over 1½-in.: Find the horizontal marked 2½ and follow it to its intersection with the 85 per cent. line, and the diagonal passing through this point corresponds to the product from a 2½-in. adjustment. By following this diagonal to the 1½-in. line, the percentage shown is 50, and subtracting this from the 85 per cent., it will be seen that 35 per cent. of the product will be over 1½ in. and under 2½-in.

Similarly the percentage of any size may be found, and it is thus possible to determine, with a fair degree of accuracy, what proportion it may be necessary to recrush or what proportion the different sizes from the plant will be.

If the material has already been screened through a certain perforation, or the percentage of a given mass is to be determined without reference to the setting of the crusher, then the corresponding diagonal should be selected on the 100 per cent. line.

A New Metal with High Heat-Resisting Qualities

Bulletin 118 of the Diamond Power Specialty Co., gives the following information regarding "Insuluminum," a new alloy recently developed in the research laboratories of the General Electrical Co. by impregnating iron and steel with an aluminum alloy. The resulting material is a rich ferro-aluminum said to be of great hardness, and possessing heat-resisting qualities much in excess of iron and steel. The process is claimed to form a homogeneous alloy which cannot be destroyed without destroying the mass of which it is a part. When tubes are coated with this material, it is said that the outer and inner surfaces of the tubes are covered with a thin film of alumina, under which is a wide band of ferro-aluminum. The alumina resists heat and is said to make the tubes much more lasting, especially in boilers or any other place where a high resistance of heat is important.

To Find the Tension in a Cableway Rope

The following quotation is from *Leschen's Hercules* of June, 1916:

"A cableway rope should be chosen so that there will be a factor of safety of four when the maximum load at the center of the span is producing a deflection equal to 5 per cent. of the distance between sup-

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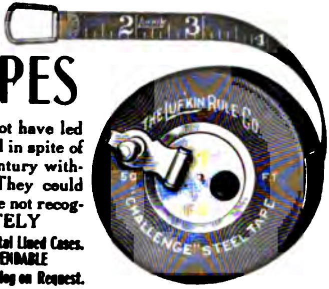

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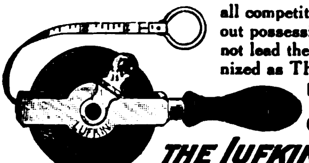

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Trade Catalogs

Bulletin 183, issued by The Jeffrey Manufacturing Co., deals with mine ventilation. It illustrates and describes the various kinds of Jeffrey ventilating fans, fan installations and fan casings. It also contains ventilating formulæ for Jeffrey fans, and tables compiled to give a comprehensive idea of the results to be obtained from the various sizes of the Jeffrey Stepped Multi-Bladed Fans.

The Ingersoll-Rand Co. has recently issued the following bulletins:

Form 9024, on Steam Condensing Plants—Beyer Barometric Type. This bulletin describes the fundamental principles of steam-condensing plants and makes comparisons between the Beyer Barometric Condenser and Low Level Jet Condensers and Surface Condensers. Auxiliary apparatus is illustrated and described in detail.

Form 4122 describes the IR Model Leyner Drill Sharpener. It explains and illustrates the sharpener in detail and shows the various styles of bits.

Form 3033 deals with the Imperial "XPV" Duplex Steam Driven Compressors, designed and constructed to operate satisfactorily under "high pressures" and "superheat" as well as under "ordinary steam" conditions. It shows the various sizes and capacities and explains in detail the operation of the Imperial Piston Valves.

Form 8112 describes the 56-H "Crown" Pick, a pneumatic hammer said to be especially adapted to stripping and picking coal in coal mining, breaking cores in foundry work, tearing up asphalt pavements and loosening and wedging out bricks and stones when demolishing old structures.

Catalog D, 1916, issued by the Denver Fire Clay Co., gives concise information regarding the Case Metallurgical Furnaces. The oil-burning furnaces are featured, particularly the Tilting Crucible Furnace, but many that are fired by gas, gasoline coal, wood or coke are also described. The last half-dozen pages of the pamphlet are devoted to crushers and pulverizers.

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Contents

Genesis of Certain Ore-Deposits. By S. F. Emmons.
Structural Relations of Ore-Deposits. By S. F. Emmons.
Geological Distribution of the Useful Metals in the United States. By S. F. Emmons. Discussion, by John A. Church, Arthur Winslow, S. F. Emmons, and William Hamilton Merritt.
Torsional Theory of Joints. By George F. Becker. Discussion, by H. M. Howe, R. W. Raymond, C. R. Boyd, and George F. Becker.
Allotropism of Gold. By Henry Louis.
Superficial Alteration of Ore-Deposits. By R. A. F. Penrose, Jr.
Some Mines of Rosita and Silver Cliff, Colorado. By S. F. Emmons.
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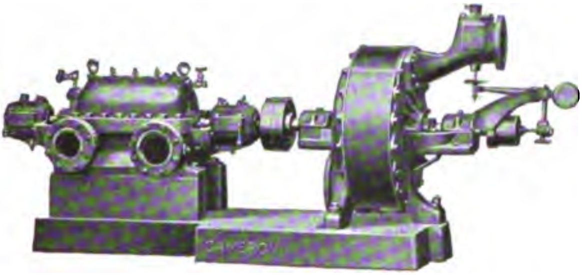

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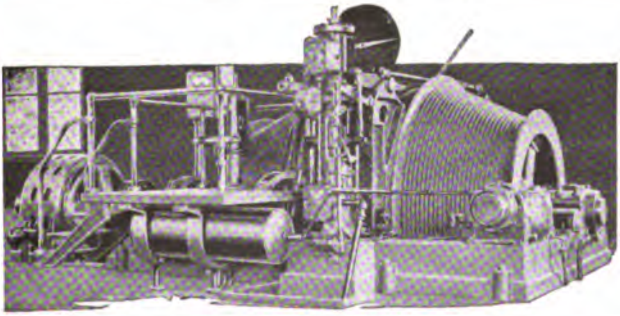

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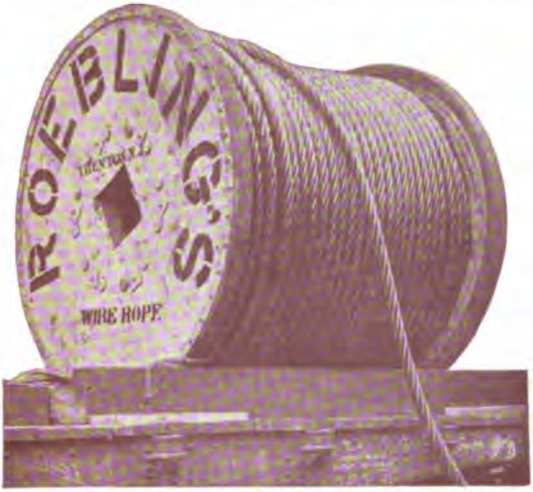

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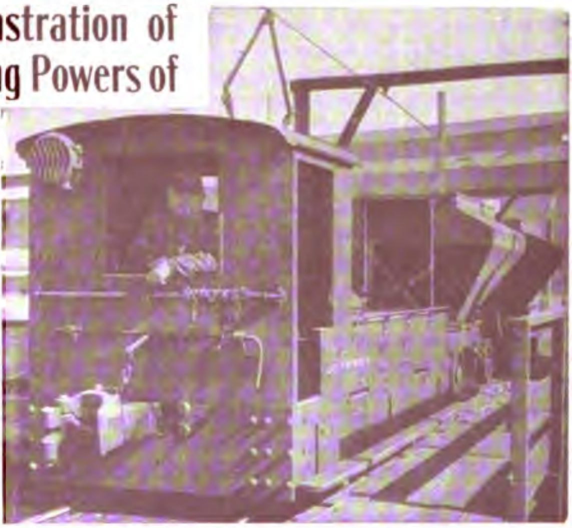

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